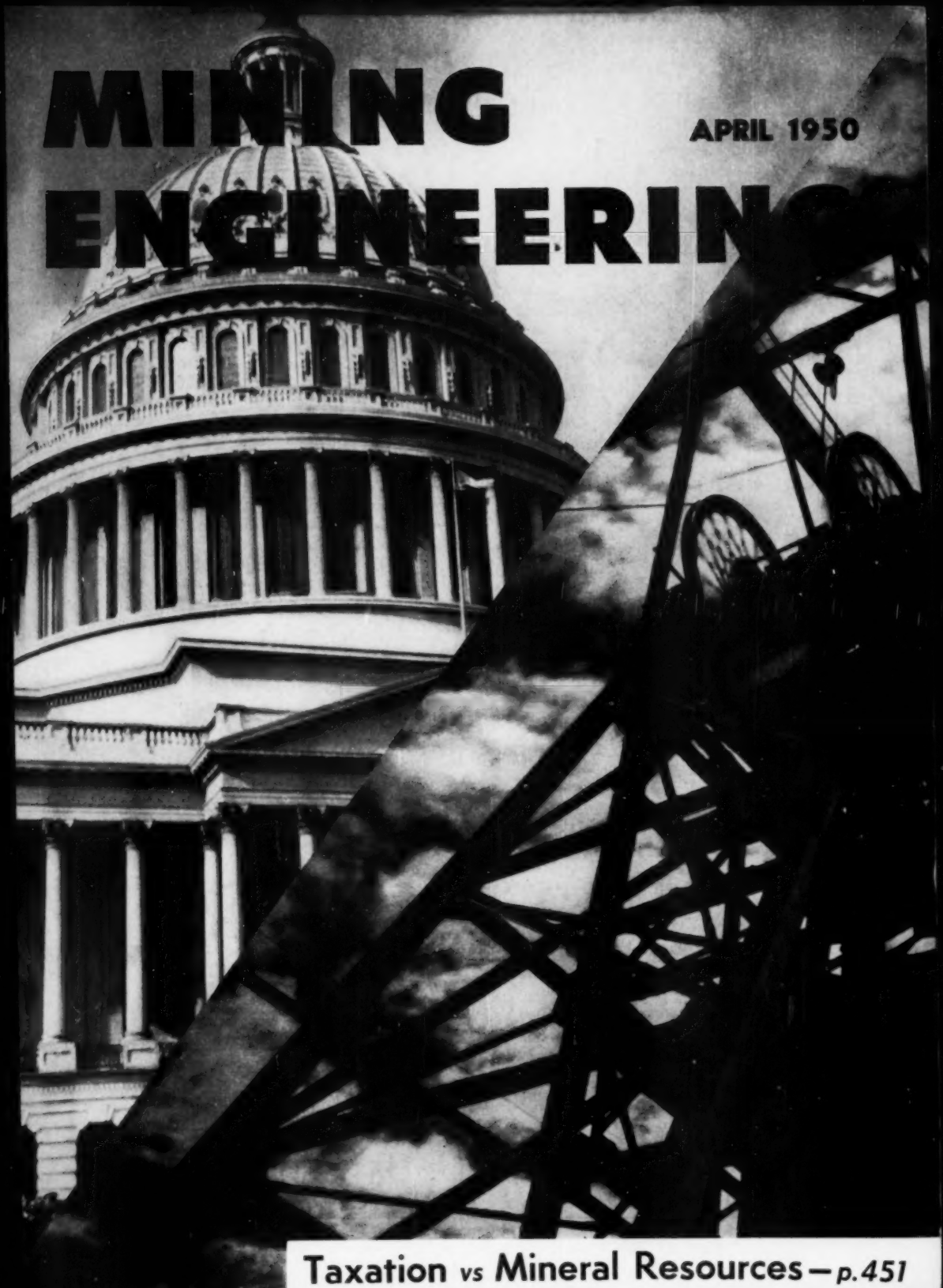
MINING ENGINEERING

APRIL 1950



Taxation *vs* Mineral Resources — *p. 451*

CRUSHED ROCK

In a recent government dam project, our engineers recommended and installed a 5 inch Overhead V Belt driven WILFLEY Sand Pump (illustrated below). This unit efficiently handles 550 GPM of minus ¼ inch crushed rock, 50 per cent solids and 55 foot head.

- Continuous trouble-free operation
- Cost-reducing performance
- Stepped-up production

In similar projects all over the world, you'll find dependable WILFLEY Sand Pumps right on the job stepping-up production, operating continuously and delivering true high efficiency, with consequent power-and cost-savings. There is an economical WILFLEY Pump size for every pumping requirement. Individual engineering on every application. Write or wire for details.

Buy WILFLEY for Cost-Saving Performance

Companion to the famous WILFLEY Acid Pump

WILFLEY
centrifugal PUMPS

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MINING TECHNOLOGY
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MINING ENGINEERING

VOL. 187 NO. 4 APRIL 1950

In This Issue

COVER

As mining engineers, we know that the ore drawn up by that mine hoist forms an important part of the solid foundation of the democracy symbolized by the Capitol Dome in Washington. We also know that government cooperation with the mineral industries is essential to our mutual goal—maximum production under the banner of free enterprise. See "Taxation vs. Mineral Resources," beginning on p. 451, for a discussion of whether or not government is playing its part.

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The Amsco® Rever-SharP

The first important reversible dipper tooth improvement made in years!

it's new . . . A reversible, self-sharpening and self-hardening tooth.

it's economical . . . Improves digging efficiency. Lasts longer—with higher resistance to breakage. Tips made of Chromanal; Shanks and Adaptors of Manganese Steel.

FEATURES OF THE AMSCO REVER-SHARP THAT LOWER HANDLING COSTS PER TON

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Unique Cutting Edges—Increase service life and cutting efficiency from 50 to 200%. Reinforced and extended corners take initial impact and work-harden faster; wear is equalized along entire cutting edge.

Special Adaptor—Made of AMSCO Manganese Steel and designed so that extra metal protects base connections and lip from wear. Available to fit all base connections.

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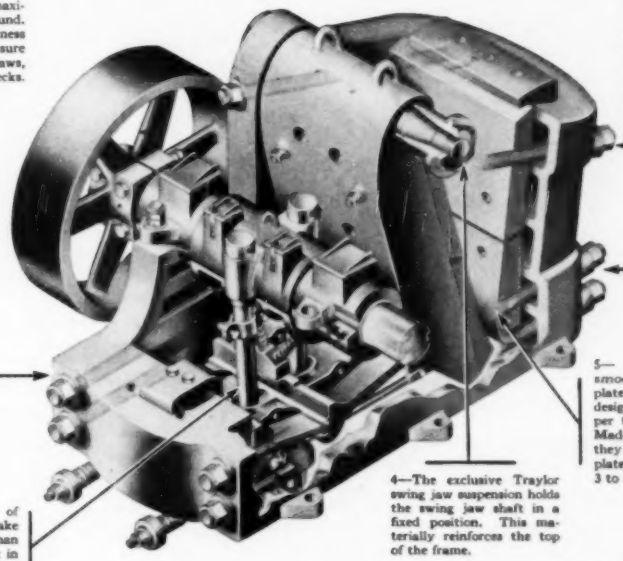
CURVED JAWS...

FOR GREATER CRUSHING EFFICIENCY IN THE TRAYLOR TYPE R JAW CRUSHER

This cut-away of a Traylor Type R Crusher shows clearly the advanced design characteristic of all Traylor-built equipment. Each feature contributes to larger profits through higher production with lower power and maintenance costs. The Type R Crusher, built in five sizes up to 56" x 72", is completely described in illustrated Bulletin 123 shown below.

1—The Meehanite* metal frame is of box section design which provides maximum strength per pound. Uniform metal thickness and annealing in sand insure castings free from flaws, shrinkage cracks and checks.

2—Heavy steel reinforcing rods are shrunk in place on each side at the top of the crushing chamber and through the strengthening band at the bottom of the frame.



3—Forged steel rods of extra large diameter make up a very strong pitman which is relatively light in weight.

4—The exclusive Traylor swing jaw suspension holds the swing jaw shaft in a fixed position. This materially reinforces the top of the frame.

5— The *non-chokable, smooth-face curved* jaw plates, of patented Traylor design, keep horsepower per ton requirements low. Made of manganese steel, they outwear conventional plates, often by as much as 3 to 1.

(* Reg. U. S. Pat. Off.)

Write today for your copy of Bulletin 1123. See how curved jaw plates can increase your profits by reducing your crushing costs.

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Rotary Kilns, Coolers and Dryers
Grinding Mills . Crushing Rolls
Jaw, Reduction and Gyratory Crushers.

A "TRAYLOR" LEADS TO GREATER PROFITS

SULPHUR

*Interesting Facts Concerning This Basic Raw Material from the Gulf Coast Region

*LOADING

Sulphur intended for vessel shipment is brought to Galveston by rail from the mines at Newgulf. It is transferred directly from cars or from storage bins to the vessel.

The loading plant consists of two parallel storage bins spaced far enough apart to allow room for railroad tracks, tracks for the hoppers and cranes, and an endless conveyor belt. The belt along the center line between the bins is straddled by four cranes and their movable hoppers.

The cranes pick up the sulphur from the bins or cars and discharge into the hoppers, which automatically feed the belt. It is weighed while on this moving belt. After weighing it is discharged onto a smaller belt which in turn discharges through a cylindrical telescopic spout directly into the vessel's hold.

Loading operations at one of the huge vats of Sulphur at our Newgulf, Texas mine. Such mountains of Sulphur are constantly being built at our mines, from which shipments are continually made.

TEXAS GULF SULPHUR CO.
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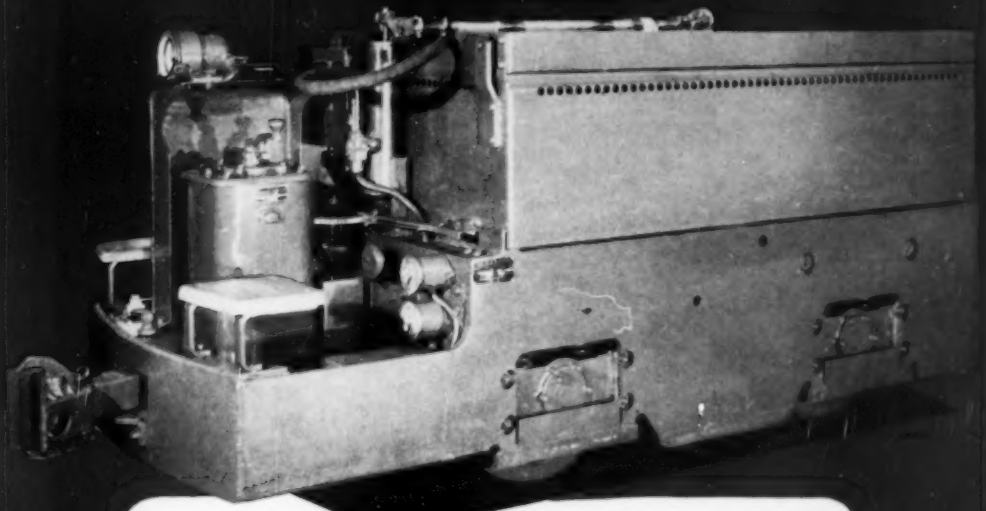
75 East 45th St. New York 17, N. Y.

Mines: Newgulf and Moss Bluff, Texas

A GOODMAN
Metal Mine LOCOMOTIVE



A two motor unit that operates on battery (140 volts) for development work and for gathering at loading chutes; on trolley (250 volts) for main line haul to shaft bottom. Battery is charged automatically when operation is on trolley.

FEATURES: directional camtactor controller; roller bearing journals; hand lever brakes; trip riders seat at cab end; offset mounting of battery box (to right or left) to give clearance at narrow Granby dump; full protection of controller, motors, resistance and batteries from water. Chassis weight is 6 tons; length without couplers —11 ft. 2 in.; height over box — 56 in.; width of chassis — 52 in.

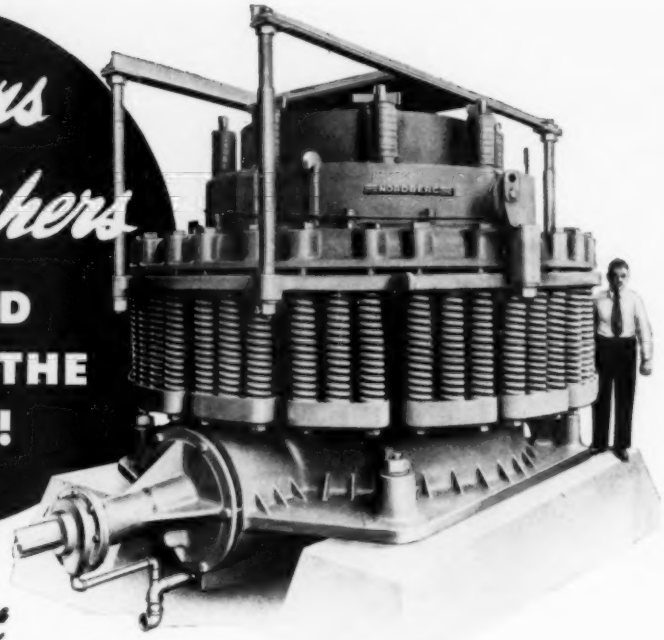
50th Anniversary 1900 GOODMAN 1950

Goodman builds a complete line of trolley, storage battery and combination type locomotives. An experienced sales engineer will be glad to help you make a selection to meet your requirements.

HALSTED ST. AT 48TH GOODMAN MANUFACTURING COMPANY CHICAGO 9, ILLINOIS

Symons Cone Crushers

ARE USED ALL OVER THE WORLD!



Here are just a few reasons why!

● *Symons Cone Crushers control their feed automatically!* The feed distributor provides an even flow of material around the entire crushing cavity . . . insuring a correct rate of feed for maximum production.

● *The crushing action of Symons Cone Crushers* differs from all conventional types of gyratory crushers by combining smashing impact, high speed and wide throw . . . timed against gravity to

PRODUCE GREATER QUANTITIES OF FINELY SIZED PRODUCTS, WITHIN CLOSER CONTROL LIMITS, WITH TROUBLE FREE OPERATION

SYMONS Cone Crushers are available in three types — Standard, Short Head and Intermediate —in a wide range of sizes to fit your requirements. Other Nordberg machinery for the mining industry includes: Primary jaw and gyratory crushers, grizzlies, screens, grinding mills, feeders, hoists and Diesel engines.

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NORDBERG MACHINERY

M501

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Machinery for processing ores and industrial minerals

WRITE FOR YOUR COPY

of Bulletin #5002, a brand new 4-page, 2 color leaflet containing pertinent data on the many features of the Dorrco Type W Pump. This rugged giant will handle up to 75 cubic feet per minute . . . has variable stroke adjustment while in operation . . . is designed for high mechanical efficiency!

THE DORR COMPANY
BARRY PLACE
STAMFORD, CONN.

☐ Yes, I'd like a copy of Bulletin #5002 describing the Type W Pump.

☐ Also send me a copy of Bulletin #5001 covering the V Type Pump.

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If you do not already have Bulletin #5001 covering the Dorrco V Type Pump, check both boxes on the coupon above. No obligation, of course.

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DORR

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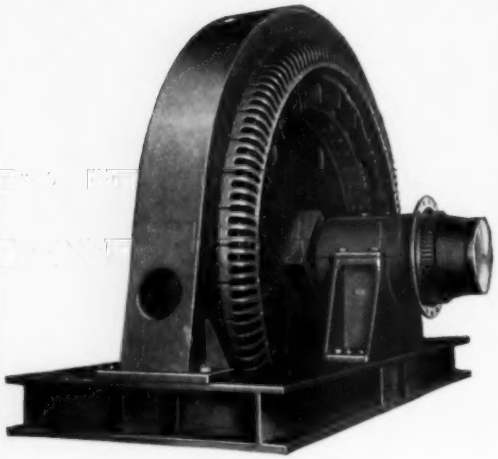
Save on original costs

use G-E synchronous

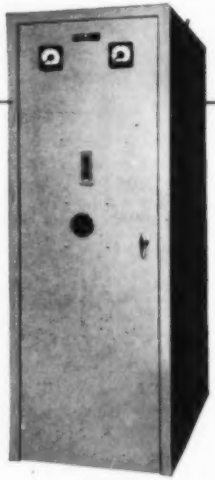


You'll save money in many applications when you specify General Electric synchronous motors. The chart shows (ratings above the black line) the applications where synchronous motors (including control and exciter) have a lower first cost than any other kind of motor. And even in cases where there is no original saving (between red and black lines) you can save on over-all cost by specifying synchronous motors to increase operating efficiency and improve power factor.

When you specify G-E synchronous motors, you get application engineering by experts in the electrical problems of *your* industry. You get the motor that gives you greatest return on your investment.

For your next large, constant-speed motor, ask your General Electric sales representative to tell you how you can take full advantage of the money-saving features of G-E synchronous motors. Or write for bulletins on your letterhead to *Section 770-17, Apparatus Dept., General Electric Co., Schenectady 5, N.Y.*

Low speed..GEA-5332 **High speed..GEA-5426**
Control..GEC-505



PROTECT YOUR SAVINGS WITH G-E SYNCHRONOUS MOTOR CONTROL

New rotor field circuit provides accurate field application—motor is protected against starting shock.

New rotor field circuit keeps constant check on motor performance and, on pull-out, field is removed before first slip cycle is completed—to protect against electrical and mechanical damage.

In addition to the protection offered by the new rotor field circuit, the **new, smaller air-break Limitamp controller** provides fast-acting EJ-2 Fuses that limit short circuits to less than one-half cycle for motors up to 5000 volts.

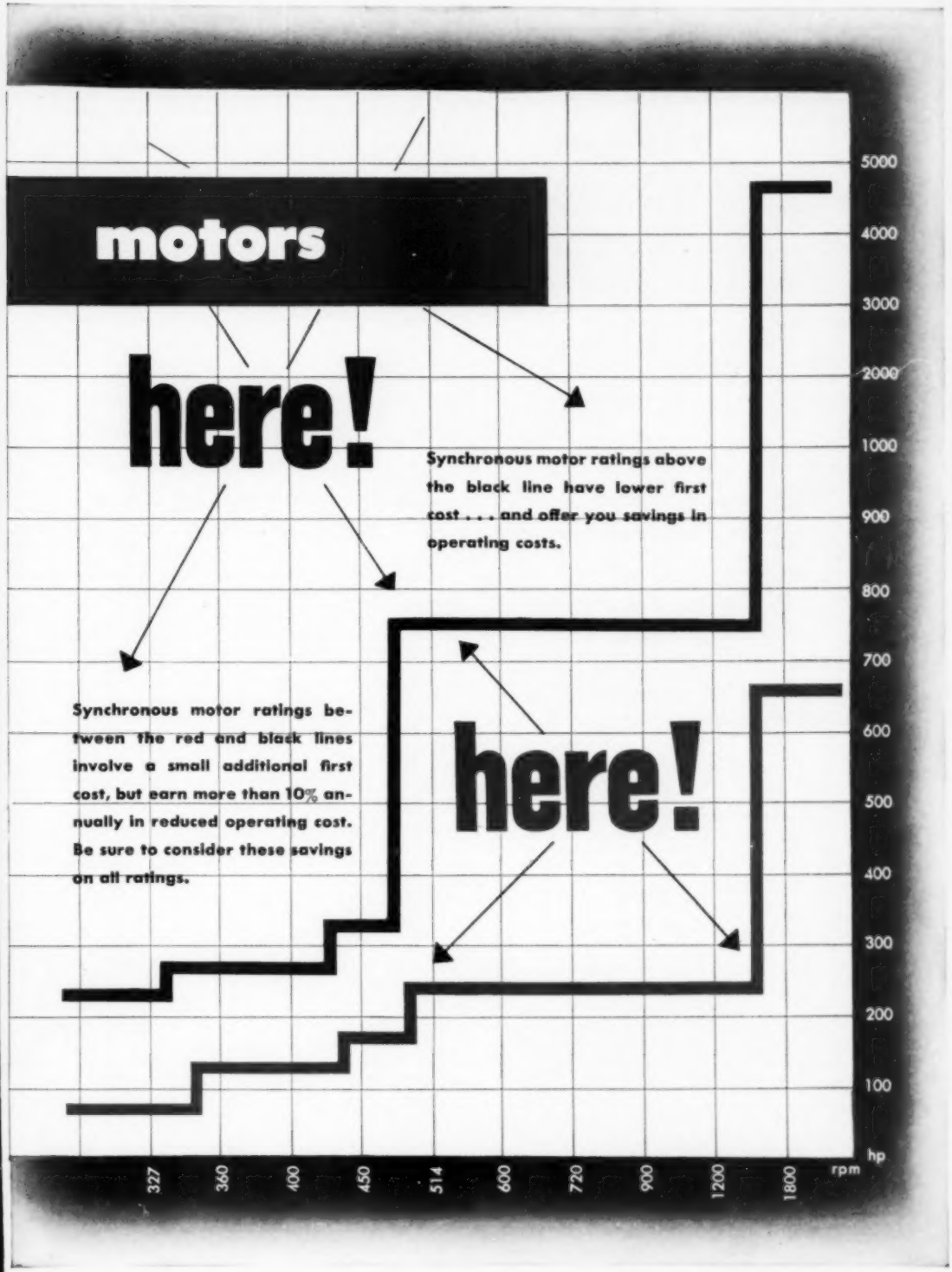
Low-voltage motors (up to 600 volts) are protected by compact, easily inspected air-circuit breakers.

To make it easier to install, to adjust, to operate, and to service, G-E synchronous control is factory-mounted in a steel enclosure with anti-glare meters and all the built-in accessories required for your application.

Come to General Electric for the engineering and manufacturing that will assure you of adequate protection of your motor and plant investment. Write for more information on this precise motor control.

GENERAL ELECTRIC

770-17


motors
here!
Synchronous motor ratings above the black line have lower first cost . . . and offer you savings in operating costs.
Synchronous motor ratings between the red and black lines involve a small additional first cost, but earn more than 10% annually in reduced operating cost. Be sure to consider these savings on all ratings.
here!
5000
4000
3000
2000
1000
900
800
700
600
500
400
300
200
100
hp
327
360
400
450
514
600
720
900
1200
1800
rpm

Personnel

Engineering Societies Personnel Service, Inc.
New York—8 West 40th St.
Detroit—100 Farnsworth Ave.
San Francisco—57 Post St.
Chicago—84 E. Randolph St.

In applying for positions advertised by the Service, the applicant agrees, if actually placed in a position through the service as a result of these advertisements, to pay a placement fee, established to maintain an efficient, nonprofit service.

All replies should be addressed to the key numbers indicated and mailed to the New York office. Include six cents in stamps for forwarding application to the employer and for returning when necessary.

A weekly bulletin of engineering positions open is available to members of the cooperating societies at a subscription of $3.50 per quarter or $12 per annum, payable in advance.

Positions Open

Assayer, single, young, preferably with foreign experience, to do fire assays, and metallurgical analyses on gold, silver, lead, zinc, etc., for Central America on two-year contract at $3000 plus room and board. Y-3088.

Division Mine Foreman, with considerable mining experience, preferably a mining graduate. Salary $325 a month with an increase to $350 at the end of three months, plus room and board for single man. Furnished apartment at end of three months if married, plus $65 per month board allowance. Location, Honduras. Y-3123.

Mining Engineer, graduate, with underground mining and supervisory experience, to act as shift boss, for underground hard ore mine using shrinkage and sublevel stopping. Salary, $3600-$4200 a year. Location, New Jersey. Y-3132.

Recent Graduate with mining engineering degree for surveying and mine efficiency work. Salary open. Location, Virginia. Y-3285.

Mill Superintendent, 30-40, with supervisory silver lead milling experience, to take charge of plant. Must speak Spanish well. Salary, $4500 plus board and transportation. Location, Central America. Y-3289

Mill Superintendent, graduate, 41, married, no children. Wide experience on mill operation, construction and maintenance. Capable, progressive, cost conscious, good record handling men. Fluent Spanish. Available short notice. M-518.

Engineers. (a) Metallurgist-Mill Foreman, graduate, experienced ore dresser and mill operator. Working knowledge of Spanish essential. Standard three years contract. Salary $5000 a year plus bonus. Single status for six months. Free transportation by air for employee and wife. Four weeks vacation yearly, free living quarters. (b) Mine Foreman, graduate, experienced metal miner. Working knowledge of Spanish essential. Three-year contract. Salary, $4200 a year plus bonus. Single status for six months. Free air transportation for employee and wife. Four weeks vacation yearly, free living quarters. Location, Bolivia. Y-3311.

Mineralogist, preferably woman, for laboratory work. Should have two to three years' experience. Salary, $4500 a year. Location, New York, N. Y. Y-3354.

Men Available

Mining Engineer, 32, married, 3 children, veteran. B.S. Mining Engineering. Experience in placer valuation; small mine and mill supervision; large copper mill operation; building, highway and railroad construction; and sales engineering of mining and industrial equipment in Arizona and New Mexico. Desire responsible engineering position with reputable firm. Available two weeks. Location preferred, United States. M-504-4912-E-8-San Francisco.

Geologist, 33, married, Master's Degree, 7½ years' experience surface and underground mapping, direction of development and exploration including diamond drilling. Available immediately. M-505-501-E-1-San Francisco.

Mining Engineer, Scot, 35, married. Fourteen years' hard rock experience gold and copper. Positions held, chief surveyor, mine foreman, efficiency engineer, mine captain. Presently located in India. M-506.

Sales Supervisor, 32, married, graduate mining engineer, Pa. registration. Four years' labor, engineering and management experience of coal mines. Four years' sales of capital mining equipment. Now employed as development engineer, and sales supervisor of coal preparation, equipment, with excellent record. Desires position as either development engineer or sales management on capital equipment with wider scope. M-507.

Design Engineer, E.M., P.E., L.S. Twenty-five years' experience design of mine and power plants, buildings, plant utilities, structures. Charge of construction of mine and oil field port, married, one child. Desires position in S.E. state, consider any location. M-523.

Mining Engineer, 45, B.S., married. Over twenty years' operating, engineering, development, examining experience, in Philippines, South America and U. S. Underground, open pit, gold, copper, nitrate, lead, zinc, silver, tin, antimony, tungsten mines. Desire direction of exploration, examination or management in Latin Americas. Fluent legal, business, technical Spanish. Fluent German. M-524.

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The Longyear Organization can help you to expand present reserves. It will supply you with drills, or do your drilling under contract.
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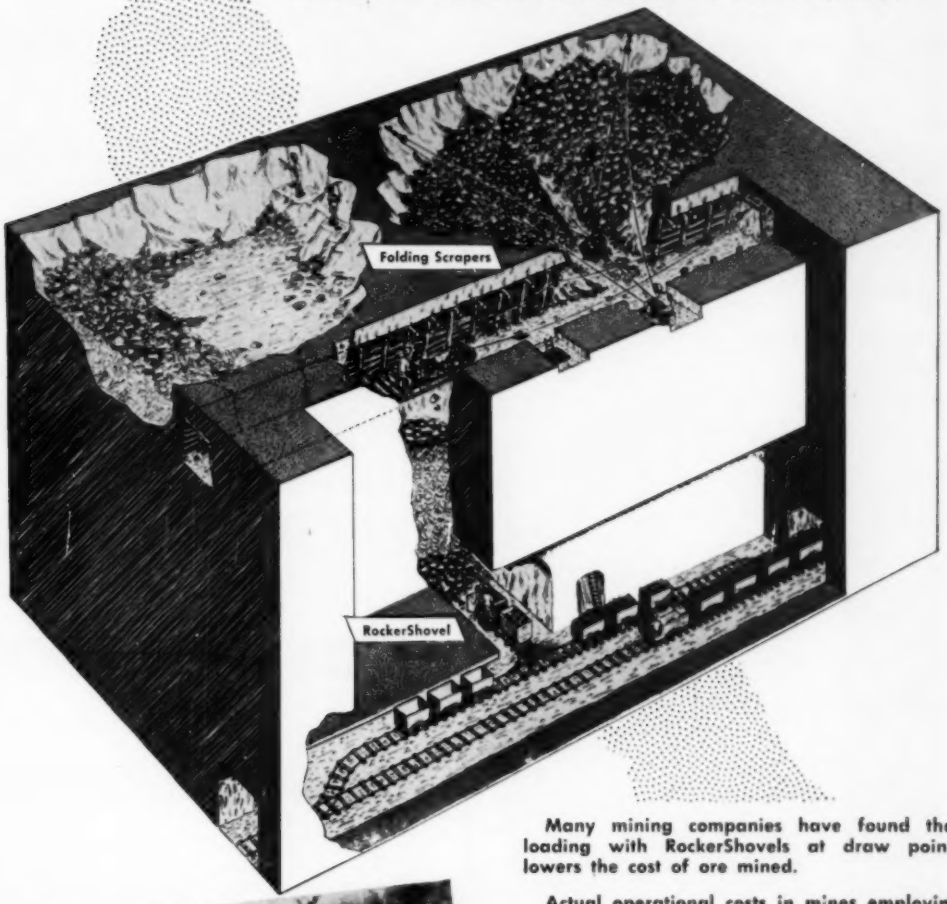
LOADING FROM DRAWHOLES with ROCKERSHOVELS



Many mining companies have found that loading with RockerShovels at draw points lowers the cost of ore mined.

Actual operational costs in mines employing both chutes and grizzles and RockerShovels show between $.08 and $.10 per ton loaded, in favor of the RockerShovels.

The main advantages, of course, are in safety. Then, there are the savings in labor required, and powder ordinarily used for secondary blasting.

Write for bulletin L-1017 on Mechanical Loading from Draw Points, showing many systems in detail.

EIMCO

THE EIMCO CORPORATION

The World's Largest Manufacturers of Underground Rock Loading Machines

EXECUTIVE OFFICES AND FACTORIES — SALT LAKE CITY 8, UTAH, U. S. A.

BRANCHES: NEW YORK, CHICAGO, BIRMINGHAM, EL PASO, BERKELEY, DULUTH

AFFILIATED COMPANIES: SOCIETE EIMCO, PARIS, FRANCE
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A228

DENVER-DILLON (Floating Circle) VIBRATING SCREENS

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SELECTIVE MINERAL JIGS

SUB A FLOTATION MACHINES

SPIRAL RAKE THICKENERS

ADJUSTABLE STROKE DIAPHRAGM PUMPS

AGITATORS CONDITIONERS

DISC FILTERS

VIBRATING SCREENS

CROSS-FLOW CLASSIFIERS

VERTICAL SAND PUMPS

AW CRUSHERS

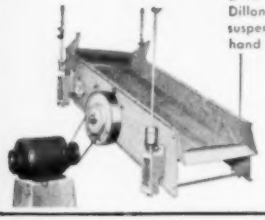
1½′ x 3′ Double-Deck Denver - Dillon Vibrating Screen, right-hand drive, showing how 1½′ x 3′ screen can be mounted on frame with casters.

1½′ x 3′ Single-Deck

Bare Screen		Add for ½ H.P. motor, motor mounting and drive.
$338	Weight 225 lbs.	Price $89 Weight 108 lbs.

1½′ x 3′ Double-Deck

Bare Screen		Add for ¾ H.P. motor, motor mounting and drive.
$371	Weight 350 lbs.	Price $101 Weight 122 lbs.



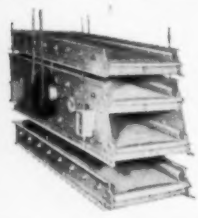
2′ x 4′ Single-Deck Denver-Dillon Vibrating Screen, suspension mounting, right hand drive.

2′ x 4′ Single-Deck

Bare Screen		Add for ¾ H.P. motor, motor mounting and drive.
$432	Weight 350 lbs.	Price $101 Weight 122 lbs.

2′ x 4′ Double-Deck

Bare Screen		Add for 1 H.P. motor, motor mounting and drive.
$473	Weight 600 lbs.	Price $117 Weight 135 lbs.



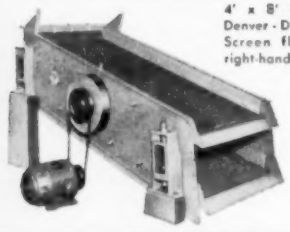
3′ x 6′ Double-Deck Denver - Dillon Vibrating Screen showing how additional deck may be added. Undersize hopper and dust or top cover may be added.

3′ x 6′ Single-Deck

Bare Screen		Add for 2 H.P. motor, motor mounting and drive.
$869	Weight 725 lbs.	Price $150 Weight 170 lbs.

3′ x 6′ Double-Deck

Bare Screen		Add for 3 H.P. motor, motor mounting and drive.
$954	Weight 1200 lbs.	Price $168 Weight 180 lbs.



4′ x 8′ Double-Deck Denver - Dillon Vibrating Screen floor mounted, right-hand drive.

4′ x 8′ Single-Deck

Bare Screen		Add for 3 H.P. motor, motor mounting and drive.
$1402	Weight 1275 lbs.	Price $168 Weight 180 lbs.

4′ x 8′ Double-Deck

Bare Screen		Add for 5 H.P. motor, motor mounting and drive.
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Authors in This Issue

G. S. Borden

GRANVILLE S. BORDEN, the happy gentlemen at right, rates a large photo in this issue. We had to include those two big fish he caught off the Golden Gate, as well as his triumphant grin. Mr. Borden has also landed a "big fish" for Mining Engineering this month—as witness his article beginning on p. 451. It's a primer for congressmen, an astute discussion of the vital tax problem facing the whole mining industry, and certainly **demands** the attention of our readers. Mr. Borden, a valuation engineer and special attorney for the Bureau of Internal Revenue, also serves as tax counsel for Standard Oil of California, the Idaho Maryland Mines Corp., the Bradley Mining Co., and Bunker Hill and Sullivan, and is a member of the bar of the Supreme Court of the District of Columbia. Mr. Borden began the hard way, as a mucker, before taking his B.S. in mining from the Univ. of California. Later he served in World War I, moving up from private to a first lieutenancy. He took his first job with the Tax Bureau in Washington while studying for his L.L.B. and L.L.M. degrees at George Washington Univ. Mr. Borden is an AIME member, lives in Piedmont, Calif., and has presented numerous other papers before the Institute. . . . **W. L. Crentz** (p. 507), coal technologist with the coal preparation section of the Bureau of Mines, and an AIME member, has been in government service since 1933, when he was a statistician with the Smokeless Coal Code Authority. Then he was with the Treasury Dept. procurement division, the Bituminous Coal Commission and Solid Fuels Admin. for War. He recently returned from an assignment in Korea on the beneficiation of Korean anthracites. Mr. Crentz was born in Baltimore, attended Eastern H.S. in Washington, D.C., and took both B.S. and M.S. degrees from the Univ. of Maryland. He has co-authored one other paper for presentation to the AIME. . . . **E. E. Donath** (p. 507), was born in Sternberg, Moravia, Germany, and took his doctorate at the Technical University in Breslau. He is now a scientific consultant for the Quartermaster Corps of the War Department and a chemist for the Koppers Co. Previously, he was chief chemist in charge of high pressure research for I. G. Farbenindustrie in Ludwigshafen, Germany. Mr. Donath is a resident of Louisiana, Mo. . . . **E. A. G. Colls** (p. 491) has been with the Consolidated Mining and Smelting Co. of Canada for 27 years, and now serves as manager of the chemical and fertilizer division there. Mr. Colls attended Marlsborough College and Sheffield Univ. He lives in Trail, B.C., where he enjoys painting and skiing in his spare time. . . . **E. L. H. Sackett** (p. 466), chief metallurgist for the Baroid Sales Division of the National Lead Co. at Malvern, Arkansas, is a 1933 graduate of the Colo. Sch. of Mines. Born in Oroya, Peru, he held his first job there, with Cerro de Pasco. He has since worked at Inspiration, with the General Engineering Co. in Salt Lake City, and in the Metallurgical department of the Howe Sound Co. in Holden, Wash. Mr. Sackett has been an AIME member since 1931.

W. L. Crentz

R. E. Bell

R. E. Bell (p. 486) has been in educational and government circles for many years, having taught at both the State College of Washington and the Univ. of Idaho, and having been with the Farm Security and Bonneville Dam administrations as both an agricultural economist and an industrial analyst. He is now associate director of the bureau of land management for the Department of the Interior. Mr. Bell was born in Pagette, Idaho, and attended high school in Raymond and Pasco, Wash., taking a B.S. from the State Coll. of Washington in 1927, and an M.S. three years later. Mr. Bell remains close to the land in his spare time, too, and enjoys camping and touring. . . . **W. W. Little** (p. 459) is mine superintendent for the United Verde Branch of the Phelps Dodge Copper Corp. at Jerome, Ariz. He began his career as a junior assayer for the U.S. Mint in Denver, became a stope engineer with the Phelps Dodge Copper Queen branch, later becoming assistant mine superintendent there. Born in Birmingham, he has an E.M. degree from the Colo. Sch. of Mines. He is also an AIME member. . . . **J. A. Ruppert** (p. 479) was born in Washington, D. C., attended McKinley High School there, and took his B.S. from the Univ. of Maryland. He has been with the Davison Chemical Co., the patent law firm of Bacon & Thomas in Washington, D. C., and is now employed as a chemical engineer with the U. S. Bureau of Mines in College Park, Md. . . . **J. E. Conley** (p. 479) is another of the chemical engineers writing for us this month. he has worked with the Bureau of Mines at Golden, Colo., in Pittsburgh, in New Brunswick, N. J., and is now senior chemical engineer for the Bureau in College Park, Md. Mr. Conley has also been with York Metal & Alloys Co. in Denver, and has done research on Wi, Mo. Ti, and U for the Westinghouse Lamp Co. He was born in New Mexico, has a B.S. in chemical engineering from the Univ. of Colorado, is an AIME member, and has presented two other TP's. . . . **F. J. Haller** (p. 475) is superintendent for the Mather mine of the Cleveland-Cliffs Iron Co., Ishpeming, Mich., and an AIME member. Born in Osceola, Mich., he attended Kenyon Coll., and the Michigan Coll. of Min. and Tech, taking a B.S. in mining. He first joined Cleveland-Cliffs in 1930. In his spare time, Mr. Haller enjoys fishing, golf, and hunting.

J. A. Ruppert

J. E. Conley

J. D. Doherty (p. 507) was graduated in chemical engineering from the University of Maine in 1926. From 1927 to 1932, he was employed by the research department of Koppers Company at Mellon Institute and at the Company's experimental units, engaged in research and development work on coal carbonization and related subjects. He was the Research Engineer of Koppers Coal Division, Eastern Gas and Fuel Associates, from 1932

J. D. Doherty

to 1945 also a member of the Technical Committee of the Coal Research Laboratory, Carnegie Inst. of Tech. and on the Technical Advisory Board of Bituminous Coal Research, Inc. In January 1945, he joined the Bureau of Mines and since 1946 has served as Assistant Chief of the Office of Synthetic Liquid Fuels, assisting in planning and directing the program being carried out in the government's five laboratories and demonstration plants for the production of oil and gasoline from coal and oil shale. This $60 million synthetic fuels research and development program is designed to equip the United States with the scientific, economic, and engineering information needed to assure long-term self-sufficiency in oil and gasoline. In 1946, Mr. Doherty visited a number of German synthetic fuel plants and arranged for the shipment to this country of selected items of equipment for use in the Government's program. This equipment included a Linde-Frankl oxygen plant with a capacity of 25,000 cubic feet per hour, which now is in operation at Louisiana, Missouri. **H. A. Schmitt** (p. 449), consulting geologist in Silver City, N. Mex., was born in Mankato, Minn., and attended the Univ. of Minnesota from 1915 to 1926, taking A.B., M.S., and Ph.D. degrees. He began his career with Standard Oil of Indiana, then worked as a junior engineer and geologist with AS&R, and spent over two years with New Jersey Zinc. Later he spent two years in Mexico for AS&R, finally becoming a consultant in Silver City in 1931. He is an AIME member, a Fellow of the Geological Society of America, and a member of the American Association for the Advancement of Science. . . . **A. F. Taggart** (p. 460) has been teaching mineral dressing up at the Columbia School of Mines for the last 31 years, and has also taught at Yale's Sheffield Scientific School. A noted AIME author, his writing career began in 1914, just four years after he graduated from Stanford, with the publication of his "Work of Crushing." At that time he was teaching at Yale, having also done examination work in Bolivia. Professor Taggart has authored chapters in the Engineers Pocket Book; the Manual of Flotation Processes (1921), the Handbook of Ore Dressing (1927), the Handbook of Mineral Dressing, and Seventy-Five Years of Progress in Ore Dressing, which appeared in the Institute's anniversary volume in 1947. "For his outstanding contribution to the progress of minerals beneficiation" he was awarded the Robert H. Richards award at the recent Annual Meeting in New York.

W. R. Webb

R. G. Fleck

W. R. Webb, who has been a research engineer for Jones & Laughlin, and production superintendent for the Blair Limestone Co., Martinsburg, West Va., is now in Star Lake, N. Y., with the J&L Ore Co. Born in Pittsburgh, he attended Allderdice High School there, and then went to Lafayette College and the Univ. of Pittsburgh, as a student of chemistry. An Institute member, he has co-authored, with H. K. Work, a paper entitled "An Experimental Open Hearth Furnace." Mr. Webb's spare time activities include fishing and golf . . . **R. G. Fleck** is also up at Star Lake, having worked with J&L in Oil City, Pa., and having been chief inspector and metallurgist with Talon, Inc. He has also worked with Republic Steel. Mr. Fleck has a B.S. in metallurgical engineering from Penn State, and he's an Institute member. Born in Niagara Falls, N. Y., he went to high school in Danville, Pa. Mr. Fleck is an-

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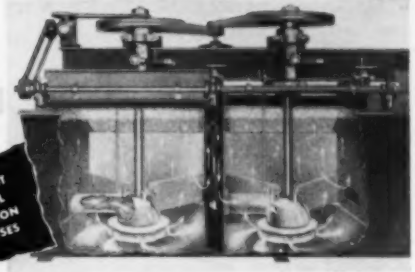

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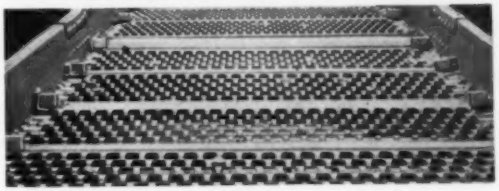

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other after-hours fishing and golf fan. . . . **John Dasher** (p. 495) is executive officer under A. M. Gaudin at the MIT engineering laboratory. He has worked as a mineral dresser with the Bureau of Mines in Alabama and Maryland, and headed the research lab for the Tin Processing Corp., Texas City, Texas. He has a B.S. in chemical engineering from the Univ. of Alabama. He has presented two other TP's for the AIME. Technology comes first with Mr. Dasher, but "television and penny-ante" fill his spare hours . . . **W. Freyberger** (p 495) is a physicist at the MIT research lab, and co-author in this issue with messrs. Gaudin, Pannell, and Dasher. Born in Newark, he attended high school in Irvington, N. J., and has a B.S. in physics from MIT. He lives in Cambridge, enjoys fishing when not on the job at MIT . . . **F. W. Bloecher** is another New Jerseyite, born in Hackensack, and having attended Wharton H.S. He has a B.S. in mining engineering from Lehigh, and took his M.S. in metallurgy at MIT last year. He has taught at Lafayette and worked with Bethlehem Steel, now functions as a research mineral engineer in the division of industrial cooperation at MIT. He co-authored an article with Prof. W. B. Plank in 1947, and was a co-author of "Radioactive Tracers" in 1948. **A. M. Gaudin** (p 495) is a noted contributor in the field of mineral dressing, having contributed between 40 and 50 technical papers to the Institute, and written "Flotation" (1932) and "Principles of Mineral Dressing" (1939). He has been professor of mineral dressing at the Univ. of Utah and at the Montana School of Mines, and was Richards professor of mineral engineering at MIT. He now directs the MIT mineral engineering laboratory, and functions also as a consulting metallurgist. Born in Smyrna, Turkey, of French parents, Mr. Gaudin attended lycees at Versailles and Toulon, France, and then came to Columbia University for his E.M. degree from the school of mines. He also holds an Sc.D. (Hon.) from the Montana School of Mines. Mr. Gaudin now resides in Newtown, Mass., enjoys painting, fishing, photography and philately in his spare time. . . . **Gerald Sherman** (p 467), who is now a consulting engineer in New York, graduated from the Columbia Univ. School of Mines back in 1894. He began his career in Grass Valley, Calif., becoming a clerk, assayer, engineer and then assistant superintendent of the Empire and North Star mines. He took charge of the mine department at the Copper Queen mine in Arizona, later becoming a consulting engineer for Phelps Dodge. He has previously presented a paper to the AIME on the operations of Copper Queen. Mr. Sherman, an AIME member, enjoys photography and traveling. . . . **J. H. Pannell** (p 495) is chief of the physics section at the MIT laboratory in Watertown Arsenal, Watertown, Mass. He has been with the Electrometallurgical Co. in Niagara Falls, and also with the Los Alamos Scientific Laboratories. Born in Brooklyn, N. Y., Mr. Pannell attended Birmingham University in England, graduating with a B. Sc. degree. He now lives in Natick, Mass.

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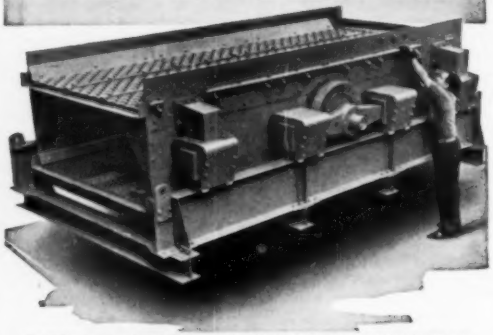

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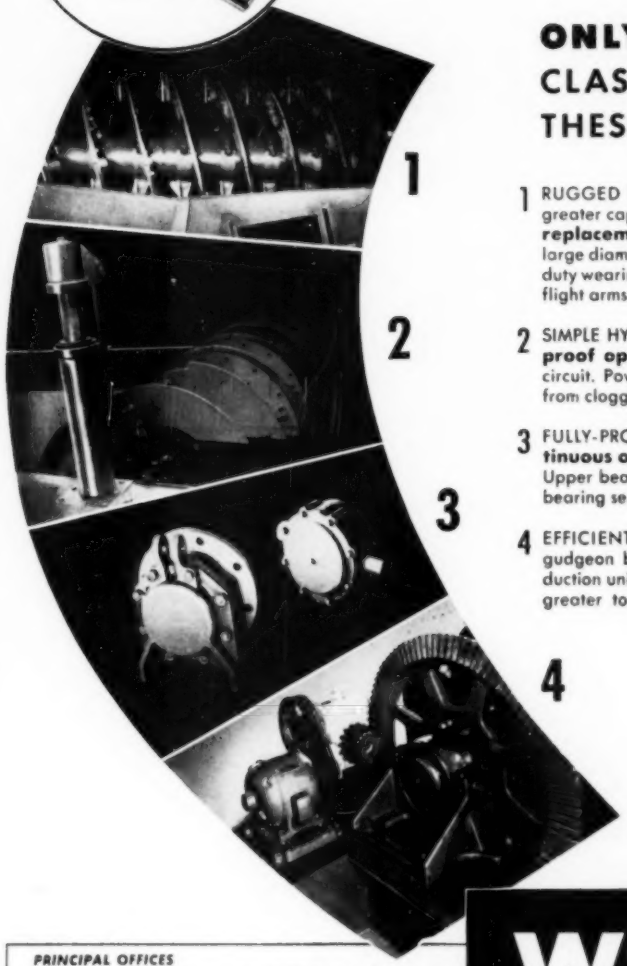

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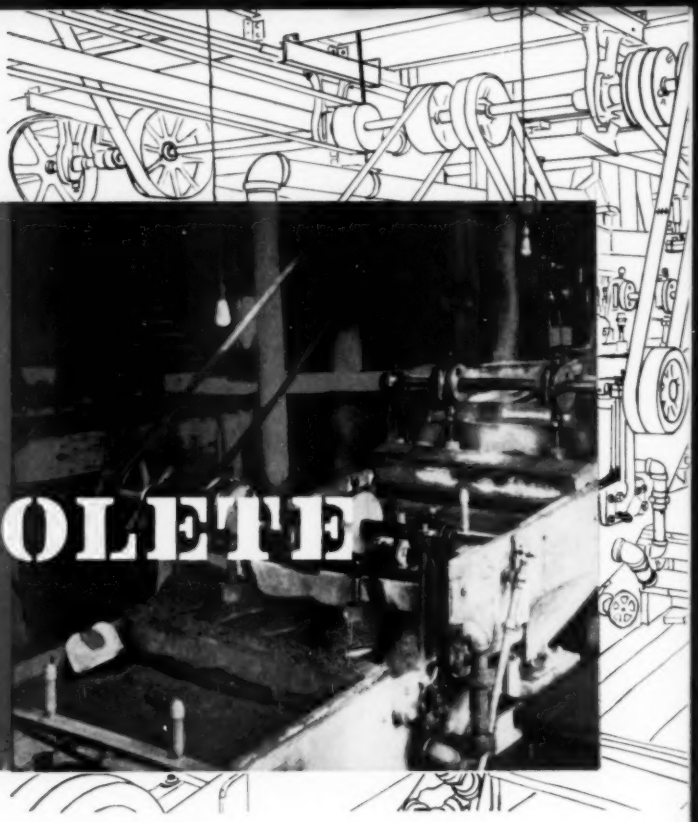

LEAD-ZINC FLOW SCHEMES GROW OBSOLETE TOO!

Read How 7 Mills Modernized Milling Practice to Increase Operating Profits

Vanners, tables and jigs served well as primary milling methods only to be superseded by more selective, more efficient chemical flotation. By the same token, traditional lead-zinc reagent combinations are themselves being obsoleted by technical progress and economic change.

Many metallurgists find that substitution or addition of the newer Cyanamid reagents produces higher grades, lower tails or both. Others find that modified combinations of older reagents in new flow schemes can offset rising costs or lower-grade mill feed. All started with an open mind . . . realizing that as the list of new reagents grows, possible combinations are as many and varied as ore bodies, operating conditions and competitive situations in the base metal industry.

Typical of what can be done are these abstracts from among the many examples of progressive base-metal beneficiation in Cyanamid Mineral Dressing Laboratory records:

ARIZONA Profitable operation of a 100 ton per day mill treating a lead-zinc-gold ore in Arizona was achieved when the correct reagent combination was found for the large percentage of oxidized lead ore in the mill feed. In obtaining a concentration of the sulphide and oxidized lead minerals, AEROFLOAT® 31, cresylic acid and soda ash are added to the ball mill. Thiocarbanilide 130 and xanthate, added to the sulphide lead rougher cells, are used as sulphide lead promoters. To the tailing from the sulphide lead cells 3 lbs. per ton of Reagent 425;0.4 lbs. per ton of xanthate and large amounts of sodium sulphide are added. From feed containing 1% sulphide lead and 10% oxide lead, a combined lead concentrate is made assaying 60% Pb with a tailing of 1.5% Pb. Overall recovery is 82%, with 16% of the lead in the sulfide concentrate and 66% in the non-sulfide concentrate.

Further testing in the field has shown that, by adding Reagent 425 to the ball mill, recovery in the sulphide concentrate can be increased to 25%, and recovery in the non-sulphide concentrate to 67%—making an overall recovery of better than 90% with no loss in grade. These results confirm other tests, tending to prove that lead recovery is increased if the slimed particles are coated with reagents as soon as formed.

IDAHO At a large sulphide lead-zinc flotation mill, extensive tests have been conducted over the past two years to determine the best reagent combination. As a result, the following combination has been adopted in place of reagents formerly used:

Pb Circuit	*Zn Circuit*
AERO Brand Cyanide	Cresylic Acid
Zinc Sulphate	Copper Sulphate
AEROFLOAT 31	Sodium AEROFLOAT "B"
Cresylic Acid	

Recoveries are improved and grade of zinc concentrate has increased from 46% to 52% Zn.

CANADA The presence of mica is not often a problem in a lead-zinc-silver ore but it made maintenance of lead concentrate grade erratic at this 550 ton per day Canadian flotation plant. Ore contains 1.2% lead; 5% zinc and 2 ounces of silver. Grade varied from 35 to 70% Pb, depending on the amount of mica floating with the lead. Tests with various traditional and new reagents showed that the use of only .04 lbs. per ton of Reagent 620 in the lead cleaner circuit would depress the mica and stabilize grade at 60% Pb with excellent lead recovery. Because the mica can now be separated from lead successfully through the use of Reagent 620, mica is a potentially marketable product, instead of a source of trouble.

ARIZONA At this lead-zinc property two classes of ore are mined and milled separately; 500 tons per day of mixed sulphide ore and 100 tons per day of oxide lead ore containing cerussite, wulfenite, galena, anglesite and vanadinite. Test work showed the best promoter combination in the oxide circuit to be Reagent 425 and Reagent 301, used with sodium sulphide in a soda ash circuit with a pH of 8.8 to 9.0. AEROFLOAT 31 as a supplementary promoter and frother, and cresylic acid as a frother are also used. From a head value of 3.23% Pb, a recovery of 88.55% was made in a concentrate assaying 50.52% Pb. Further testing on the mill showed that changing the point of addition of the AEROFLOAT 31—cresylic acid mixture from the first flotation machine to the grinding circuit resulted in increasing recovery of lead to 93%. Recoveries of precious metal values are excellent.

In the sulphide lead-zinc circuit AERO Brand Cyanide has been found most effective as a depressant in the lead circuit, with Thiocarbanilide 130 as the principal lead promoter.

COLORADO A large flotation mill in Colorado treating complex mine and custom ores containing lead, zinc, gold, silver, and copper, mainly in sulphide state, found that special reagents could improve operating results. In this mill, a bulk float of sulphide minerals is first made with Reagent 404 (.08 #/Ton) and Pentasol Xanthate. The bulk concentrate is reground at a pH of 8.5 with AERO Brand Cyanide, lime, zinc sulphate to depress the zinc. Sodium sulphite and a small amount of Reagent 444 are added to float a concentrate containing gold, silver, lead and copper which is shipped to the smelter. The zinc minerals are activated at a pH of 9.4 with copper sulphate and floated with Sodium AEROFLOAT "B" for shipment to the zinc smelter.

AFRICA A 1200 ton per day lead-copper-zinc producer in Africa is treating approximately 500 tons of highly oxidized dump ores containing sulfide and oxide minerals in one section of the mill and 700 tons of run of mine sulfide ore in separate grinding and flotation circuits. Dump ore flowsheet involves first the production of a bulk sulfide concentrate, then the sphalerite in this concentrate is floated away from the lead and copper sulfides using sodium dichromate and approximately 0.25 lbs. per ton of ore of Reagent 610. Satisfactory grades of zinc concentrate and lead-copper concentrate are produced.

The oxide lead and copper minerals in the bulk sulfide tailing are floated with sodium-hydrosulfide, Reagent 301 and varying amounts of Reagent 404 up to 1.0 lb. per ton of ore. Run-of-mine ore is treated by conventional selective flotation procedures.

COLORADO This 500 ton per day base metal mill treats a sulphide ore containing galena, chalcopyrite and sphalerite in a quartz gangue. Copper and lead are floated first, followed by a zinc float; the copper and lead being separated in a third circuit. Promoters for the first float are Reagent 404 (0.05 lbs/ton) and amyl xanthate (0.04 lbs/ton). Sodium cyanide, zinc sulphate and sodium sulfite are the chief depressants. For separating the copper-lead concentrate only sodium cyanide (0.40 lbs/ton) and sodium sulphite (0.08 lbs/ton) are used. Lead concentrates average over 67% Pb, with 91% recovery from a head of 2.73% Pb. The copper concentrates average 25% Cu, with 82% recovery from a head of 0.97% Cu. The metallurgist for this operation reports that Reagent 404 was found to have the highest selectivity for galena of the many reagents tried.

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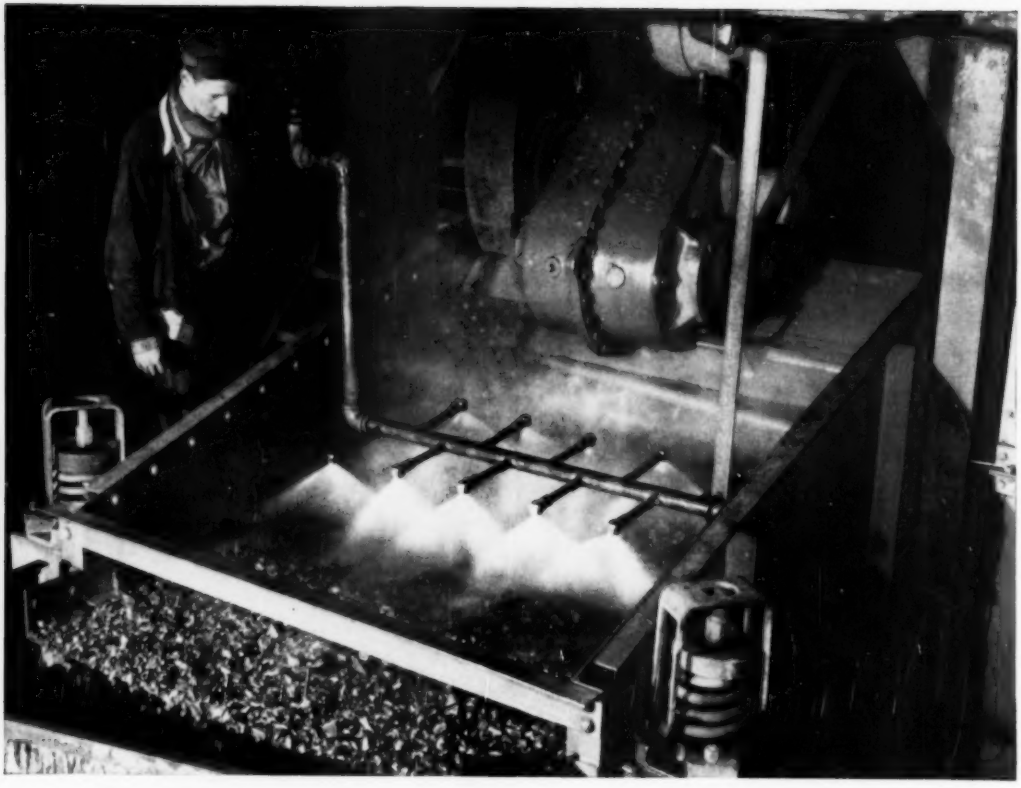

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Letters to the Editor

. . . I would like to call your attention to a rather minor matter in connection with the editing which was done on my article "Modern Electric Mine Hoists" in the November Mining Engineering. In three separate places the words "Ward Leonard" as contained in the manuscript were rewritten "Ward-Leonard". For many years I have been carrying on a one man endeavor to prevent this particular little error. This control system was first suggested by Mr. Ward Leonard, an early pioneer in electrical engineering. There is no more reason to hyphenate his name than there is Thomas Edison or Abraham Lincoln or John Beall, etc. Perhaps you could further this cause a little and correct a false impression among your editors and others by giving a little space to the matter in one of your issues.

Russell S. Sage, Engineer,
Mining Division
General Electric Co.

Encouraging New Members

As a Junior Member of the AIME I am heartily in agreement with the views expressed by Mr. Robert C. Stephenson in his letter to you which appears in the February Mining Engineering. My experience along this line has been somewhat as pointed out by Mr. Stephenson. Only recently I attended an all-day meeting of the Pittsburgh Section, and outside of a half a dozen fellows I met at the luncheon no effort was made to acquaint the younger members and visitors with the friendliness and good fellowship such a meeting should generate. It seems to me that our organization should make a greater effort to encourage attendance of the younger members at AIME meetings.

I sincerely believe that if Mr. Stephenson's suggestion to have each established member take one or more younger members under his wing at Institute meetings were adopted it would be very much to the advantage of the AIME as a whole.

I believe too that the younger members should take a more active part in the affairs and meetings of the Institute, especially at the Section level. They should also be more aggressive and prompt in accepting the responsibilities of membership, the principal one being attendance at all Institute functions. Unfortunately this, in my opinion, is one of the hardest things to do, for as Mr. Stephenson pointed out it is usually only the top brass that are in a position to attend.

In closing I would like to add my appreciation for the fine work that is being done by the staff of Mining Engineering in bringing out what in my humble opinion is the finest technical magazine in the mining field.

Luis Jordan, Jr., Chief Engineer
Rail & River Coal Co.
Bellaire, Ohio

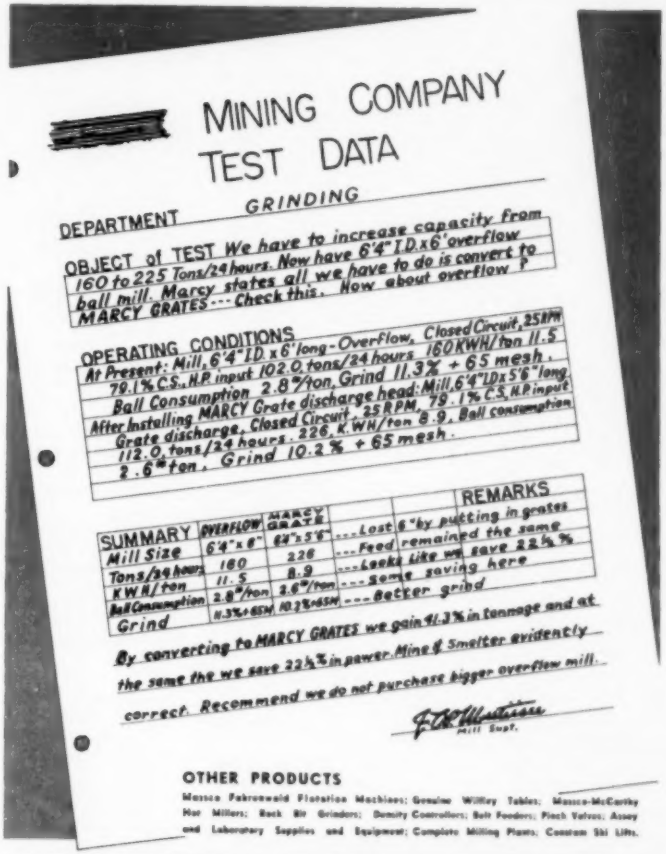
CERRO BOLIVAR

Saga of an Iron Ore Crisis Averted

The first story of U. S. Steel's "Mountain of Iron" in Venezuela, plus a fine analysis of the iron ore situation, has been reprinted in full color, and copies are available on order from Mining Engineering.

Individual copies ..... 25 cents

100 copies $17.50


MINING COMPANY
TEST DATA
DEPARTMENT GRINDING
OBJECT of TEST We have to increase capacity from 160 to 225 Tons/24 hours. Now have 6'4" I.D.x6' overflow ball mill. Marcy states all we have to do is convert to MARCY GRATES --- Check this. How about overflow ?
OPERATING CONDITIONS
At Present: Mill, 6'4" I.D. x 6' long - Overflow, Closed Circuit, 25 RPM
79.1% C.S. H.P input 102.0 tons/24 hours 160 KWH/ton 11.5
Ball Consumption 2.8#/ton, Grind 11.3% + 65 mesh.
After Installing MARCY Grate discharge head: Mill, 6'4" I.D x 5'6" long
Grate discharge, Closed Circuit 25 RPM, 79.1% C.S. H.P input
112.0 tons/24 hours. 226. K.WH/ton 8.9. Ball consumption
2.6#/ton. Grind 10.2% + 65 mesh.
SUMMARY
OVERFLOW
MARCY GRATES
REMARKS
Mill Size 6'4"x 6" 6'4"x 5'6" ...Lost 6"by putting in grates
Tons/24 hours 160 226 ...Feed remained the same
KWH/ton 11.5 8.9 ...Looks like we save 22½%
Ball Consumption 2.8#/ton 2.6#/ton ...Some saving here
Grind 11.3%+65M 10.2%+65M ...Better grind
By converting to MARCY GRATES we gain 41.3% in tonnage and at the same the we save 22½% in power. Mine & Smelter evidently correct. Recommend we do not purchase bigger overflow mill.
Mill Supt.

OTHER PRODUCTS

Massco Fahrenwald Flotation Machines; Genuine Wilfley Tables; Massco-McCarthy Hot Millers; Rock Ore Grinders; Density Controllers; Belt Feeders; Pinch Valves; Assay and Laboratory Supplies and Equipment; Complete Milling Plants; Constam Ski Lifts.

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THE **Mine & Smelter SUPPLY COMPANY**

Manufacturers News

The Nuclear Division of the Clarkstan Corp., Los Angeles, announces the Model No. 501 Alpha - Counter, a nuclear instrument for the detection of alpha particles (rays) from any source. All uranium and thorium ores, say the manufacturers, may be quickly, qualitatively, and quantitatively analyzed, when the radioactive content is as low as a fraction of 1%. The device is stable in operation so that it may be calibrated against known sources and samples. The Alpha-Counter utilizes a highly efficient optical system and a newly-developed phosphor screen. From these it provides a statistical count of alpha particle scintillations. Unlike the spintharascope, which must be used in the dark, the Model 501 Alpha-Counter may be used under subdued white light or red light illumination. The Clarkstan Alpha-Counter has applications for the prospector, assayer, teacher, and mineralogist. It comes complete with luminescence-quenching filter, radioactive samples, and aluminum carrying case.

Deflecting Wedge. E. J. Longyear Co. announces a new booklet on "Deflecting Diamond Drill Holes with the Improved Hall-Row Deflecting Wedge." Containing 28 pages and 14 charts, the text gives precise instructions for deflecting drill holes. The subject is practically treated, and is based upon Longyear experience in the field.

Gyratory Crusher. The Traylor Bulldog gyratory crusher is built in thirteen sizes. Each size quickly nips and crushes any rock that enters it. Bulletin 4100, published by The Traylor Engineering & Mfg. Co., Allentown, Pa., contains a complete description of this crusher.

Diesel Engines. Four new bulletins—166, 167, 167-A, and 168 have been issued by the Nordberg Mfg. Co., Milwaukee, on its new one-cylinder 4½ x 5¼ in. Diesel engine. These 4-cycle, 1-cylinder Nordberg Diesel engines are rated at 15 hp at 1800 rpm and 10 hp at 1200 rpm. The units are built to provide a dependable source of power over a wide range of applications. Specifications, detailed information, and outline drawings of Nordberg Model 4FS-1 Diesel engines as adapted to power take-off and without clutch (167 and 167-A), generating units (166) and pumping units (168), are given in these bulletins.

Compressors. The Joy Mfg. Co., Pittsburgh, has published a new 16-page bulletin, A-56, on their WL-80 UNITAIR compressors. These two-stage, air-cooled compressors are furnished as complete motor-driven units with three standard electric drives including built-in motor, direct-connected and V-belt drive. They may also be equipped with gas or Diesel engine. The UNITAIR has application in factories, foundries, mills, and plants.

Turntables. Bulletin No. 48 describes the Hardinge line of pressed steel turntables, which are designed for any material-processing or industrial operation where rail cars are used.

Flotation Machines. How Denver "Sub-A" flotation recovers and cleans waste coal fines at a total cost of 20 to 50 cents per ton of recovered coal is reported in a new bulletin published by Denver Equipment Co., manufacturer of Denver "Sub-A" flotation machines. Bulletin F10-B29 illustrates what happens when coal fines and ash materials are conditioned with kerosene and fuel oil and then subjected to subaeration. Typical flow sheet, preliminary laboratory testing, sources of feed, equipment required to recover coal fines, and results of laboratory tests made on samples of coal received from many parts of the world are dscribed in this bulletin. Address your request for a copy to the manufacturer at 1418-17th St., Denver 17, Colo.

Belt Conveyors. Engineering Data Book ID-481, "Continental Belt Conveyors," has been published by Continental Gin Co., Birmingham, Ala. This illustrated 64-page catalog includes engineering and application data along with dimensions and list prices.

Speed Reducers. Engineers and designers can select Falk parallel shaft sleeve bearing speed reducers quickly and easily by referring to Falk Bulletin 1110.

Dragline Scrapers. A new catalog, No. 19, describing various techniques of storing and reclaiming bulk material by means of power drag scrapers has been issued by Sauerman Bros., Inc., Chicago. This 36-page booklet is one of a series that makes up the 200-page sectional catalog covering the complete Sauerman line of scraper and cableway equipment. It contains nearly 100 illustrations of material-handling problems with explanations of their solutions. These problems include open and indoor storage of such diverse materials as sawdust, fine chemicals, sand, coal, crushed stone, fertilizers and heavy ores. Illustrations and six pages of specifications explain the details of the component parts of the scraper equipment.

Martin Machine Co., Kewanee, Ill., has added another development to their line of "Carryhaul" trailers. It is a patented folding gooseneck, which, when lowered to the ground, forms a loading ramp.

Alloy Steel. "For Longer Wear, Less Repair," a new 36-page brochure describes Jalloy, a special alloy steel manufactured by Jones & Laughlin Steel Corp. The manufacturer states that Jalloy is being used in the mining, quarrying, and earth-moving industries with outstanding success where abrasion and impact resistance are the major factors in the service life of steel.

Magnetic Equipment. "Dings Certified Magnetic Equipment" is the title of a new 8-page illustrated bulletin, C-5000A, recently released by Dings Magnetic Separator Co., Milwaukee. The bulletin describes Dings electric and nonelectric magnetic pulleys, nonelectric magnetic drums, triple pole rectangular magnets, lifting magnets, nonelectric plate magnets, high intensity induced roll and cross belt magnetic separators, and several other types of equipment. Typical uses of this equipment include removal of tramp iron to protect crushers, grinders, and other machinery; purification of magnetic minerals from nonmagnetic products.

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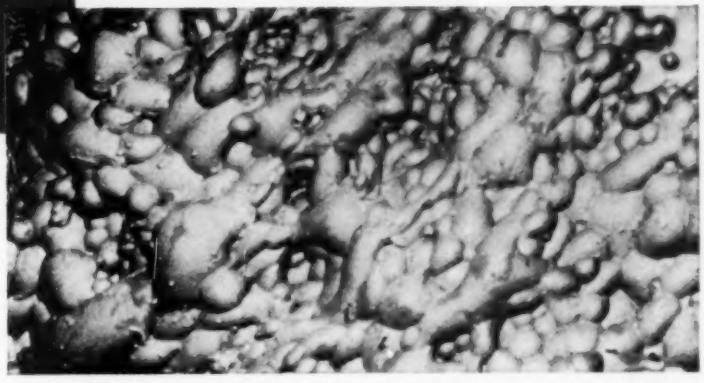

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DOW CHEMICALS

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Ground 973,617 Tons of Nickel-Copper Ore at Copper Cliff Concentrator in 27,553 Hours

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This set of Ni-Hard* liners, weighing 52,605 lbs. remained on the job, day in and day out, from November 3rd 1945 until July 27th 1949, first grinding 223,763 tons in rod mill service and then 749,854 tons in ball mill service. The weight of the scrap liners was 7365 pounds, 14% of the original weight.

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THE INTERNATIONAL NICKEL COMPANY, INC. 67 WALL STREET NEW YORK 5, N. Y.

* The Venezuelan Trade Mission to the United States announced that Bethlehem Steel Corp. will begin exporting iron ore from its El Pao concession, south of the Orinoco River in Venezuela, in about three months. The ore will be transported by rail from the mines to barges on the Orinoco and transshipped to ore vessels at the coast for shipment to the Sparrows Point, Md., plant.

The Latin American business men warned that projected increases in the United States tariff on petroleum imports from Venezuela would force that country into bilateral trade agreements with other countries and would mean drastic curtailment of Venezuelan imports from this country.

* Bethlehem, South American Gold and Platinum Co., and U. S. Steel are investigating the possibilities of securing American foodstuff for their South American mining camp commissaries through the International Basic Economy Corp. IBEC buys in quantity to supply its subsidiary companies operating in Venezuela and Brazil. The mining companies will rid themselves of this burdensome task as well as realizing a saving from the bulk purchases of IBEC.

* Oliver Iron Mining Co. has begun construction of a fine ore and taconite agglomeration plant at Virginia, Minn. It is expected to be ready in six months. The plant will be used to experiment with the making of clinker cakes and egg-sized lumps by sintering machines and nodulizing kilns respectively; about 1 million tons per year will be produced.

* Roof pinning has been adopted at the iron mines of TCI, near Birmingham, to support overlying strata as mining progresses. Favorable results have been obtained in that dilution of the ore has been reduced by 50% which is attributable to this method of supporting the roof.

* An American mining engineering team has gone to Turkey for a two-year assignment assisting Turkish government agencies in the $45 million ECA development, mechanization, and construction of cleaning plants for the Zonguldak coal field, 200 miles east of Istanbul on the Black Sea. The group consists of James Beeley, John E. Good, John S. Lambright, David Renshaw, and Charles R. Weir, all of the Paul Weir Co.

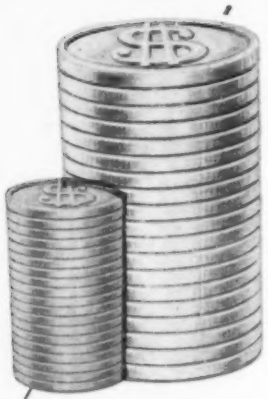
* Arthur F. Taggart describes a new way of training engineers by emphasis on broad, basic engineering courses on page 460 of this issue.

* Lawrence T. Tierney, chairman of the National Bituminous Coal Advisory Council, released a report stating that the increase in oil imports since the latter part of 1948 has caused significant damage to the U. S. fuel economy. It has reduced the strength of and employment opportunities in the coal, domestic petroleum, and transportation industries, as well as in related service and supplier fields.

* Women, camels, and donkeys are the beasts of burden of the Sahara, according to the UNESCO World Review.

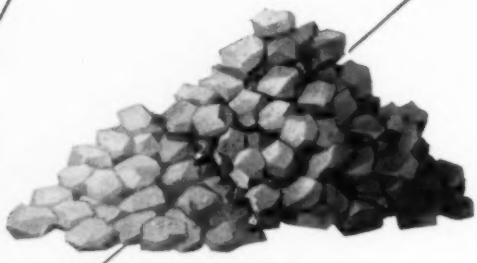
* Increased demand for magnesium sheet in airplane construction and other phases of the national security program have led Alcoa to establish a sheet magnesium rolling division at its New Kensington, Pa., works which will begin operation shortly.

* Novel prospecting methods being used by the USGS in its search for uranium include: shock-mounted instruments in light trucks, Geiger-counter probes lowered into augur holes, and experimental airborne prospecting.



GREATER SAVINGS

BETTER BREAKAGE



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XR50-6

Education Needed to Stem Adverse Taxation

ABOLITION of percentage depletion deductions for the exhaustion of mineral producing properties, as recommended by President Truman in his tax message to Congress, is a serious threat to the prosperity and military security of the nation. Venture capital for exploration and development will be a scarce item if this recommendation is enacted on top of existing "decentives" inherent in the tax structure affecting mineral properties now. In the article "Taxation vs. Mineral Resources," by Granville S. Borden, on page 451 of this issue, these decentives and their effect are set forth in detail.

In comparison with other industries, the mining industry has an unfavorable climate in which to present its case in Washington and, upon occasion, before the legislatures of the several states. This is evident when the various special benefits such as farm subsidies, cooperative exemptions, and income tax exemptions for labor unions are considered. Public inertia toward legislation affecting the mineral industries has permitted political capital to be made on important issues affecting them. Development of strategic mineral supplies has suffered. Progress in the exploitation of taconite has been impeded by taxatien upon reserves. The lethargic reception by the press of the gigantic undertaking by the U. S. Steel Corp. in the Cerro Bolivar enterprise and the concurrent taconite program are indicative of the attitude of the public toward the mining industry. In this latter instance, a stroke of triple impact upon the American people was treated in cavalier fashion. The Corporation demonstrated the daring and ingenuity of free enterprise, it provided the iron ore necessary for continuing national prosperity, and it strengthened the military security of the nation by the added reserves of that mineral.

The average citizen knows little of the problems of the mining industry. It is therefore the duty of the mining profession to take steps to remedy the situation in general, and particularly we can begin by familiarizing ourselves with the predicament of the industry imposed by our present tax laws. Mr. Borden's article is an excellent starting point. We can answer publicly-made indictments directed at mining profits at every opportunity. It is not too much to ask—considering the gravity of the situation—that we make ourselves available to address civic and social organizations on these matters.

The day when the miner, or any other enterprising individual for that matter, could stand alone and say "to hell with public opinion, what's mine is mine" is a memory of the evanescent past. We must not be loath to adjust ourselves to the changing times but must bare our travail as well as our conquests to the people.

It's Everyone's Business

MARCH 15—Industry is rapidly snapping back from another coal crisis, other business news is in general favorable and the outlook through the Spring is by most observers considered quite promising. Most industries report comfortable backlogs of unfilled orders, department store sales in recent weeks have equalled in dollars those of the corresponding period last year, retail sales of automobiles show again over the same period a year ago, and new orders for machine tools are at the highest figures since August 1946.

Orders for railway freight cars have shown a very encouraging pickup, with a total of 9385 cars in January and the N. Y. Central alone ordering 4500 in February. Construction continues very active. In January new housing starts totaled 80,000 against 50,000 a year ago and the daily average contract awards, according to the Dodge reports, were 57 pct larger. In the first 22 days of February the increase in awards was 27 pct. Lumber orders since Jan. 1 have run 22 pct above last year.

All of these favorable factors bolster the opinion that the upturn in industrial output and employment which began last summer has not yet spent itself and that the setback caused by the strikes will be made up when people get back to work. Only in a few lines are there indications that primary production may be outrunning the consumption of finished goods, although inventory build-up would seem to be rather modest as the National Association of Purchasing Agents reported in February that 78 pct of its members were buying for 60 days or less.

Industrial prices as a whole show no significant trend despite the continued alarm over both the immediate and the long-term possibility of inflation caused primarily by the government's increasing expenditure and its inability to balance the budget. A recent advance of 1.9 pct in primary market prices is viewed by some people as the surface symptom of renewed inflation, but even so it is more than two years since the all-time high was reached by the daily index of sensitive commodities in November 1947. Over two years have passed since prices received by farmers hit their peak and about a year and a half since all wholesale prices and the consumer price index touched their highest points in August 1948.

The decline in the wholesale price index has been almost continuous, interrupted only twice by slight advances in the monthly figures from February to March 1949, and, later in the year, from August to September. Of course, this continued decline in the last half of 1949 largely reflects a downward movement in farm and food prices. Wholesale prices for commodities other than farm and food leveled out after June and now stand only slightly under the June level. Farm products are down by 22 pct from January and by 18.6 pct from August 1948. Food prices are down only slightly less. Consequently, the farm price support program is in serious trouble and has imposed a heavy financial burden on the Treasury.

In those commodities other than farm products only building materials, chemicals and textiles have come down as much as the average, while housefurnishings and metals and their products have fallen hardly at all.

Glancing away from the domestic front, the results of the British general election indicated a highly inconvenient situation, that public opinion there is so exactly deadlocked as to divest the victors of much ability to make forceful decisions or create significant policy. It is generally considered that another election in the near future would give substantially the same result. The British people are equally divided and the politicians are not likely to precipitate another appeal to the country until there is manifest reason to believe that external and internal circumstances have changed sufficiently to permit one party or the other to obtain a decisive verdict.

Because of the lack of a clear mandate steel nationalization will likely lag in Britain. In the meantime there is considerable discussion going on as to whether the European steel industries as a whole are at a disadvantage in world competition with the American industry. An article in the "Statistical Bulletin" for January of the British Iron and Steel Federation takes issue on this point with the authors of the Geneva report, observing that they wrote before devaluation, and this particular conclusion "has been overtaken by events." The "Bulletin" also contends that the statement was in any case incorrect so far as Britain was concerned, and that it was based on scanty and insufficient evidence. To support the rebuttal a detailed and extensive comparison is made of the home prices of certain steel products in Britain and in other countries of the world. The accompanying table gives a selection of the prices published. It sets out the changed position since devaluation and since the recent increases in American steel prices. These are home market prices; no doubt in competitive export business some of the price levels could be brought closer together at need.

Comparison of Domestic Steel Prices in Various Countries

	Angles* 5" × 3" × ⅜"			Plates* 20' × 5' × ¾"			Heavy Bars* 4" dia.			Rails*† 90 lb. per yd. 60' lengths			Sheets‡ 6' × 3' × 20 SWG		
	£	s.	d.	£	s.	d.	£	s.	d.	£	s.	d.	£	s.	d.
UK	19	13	6	21	14	6	21	13	6	19	2	6	28	6	0
USA	33	4	0	34	0	0	36	0	0	29	0	0	39	12	0
Germany	21	11	1	22	10	6	21	0	0	21	8	6	32	10	7
Belgium	23	16	1	25	18	11	23	17	6	25	8	0	...		
Canada	30	18	2	30	18	2	32	0	0	...			38	3	8
Sweden	26	9	3	...			28	0	3	...			33	13	8
Australia	18	5	0	19	5	0	18	15	0	17	12	6	37	0	0

* 5 tons of a size; 20-ton lots. † 500 tons of a size. ‡ 25 tons of a size.
Source: British Iron & Steel Federation.

While Mr. Atlee barely squeezed through in England and King Leopold just made it in Belgium, over in Moscow Mr. Stalin got another clear mandate from his people. The "Ja" vote was overwhelming, a result which apparently was rather generally anticipated by major Russian newspapers.

A week or so prior to his trial by vote, Mr. Stalin bade a fervent goodbye to China's Mao Tse-tung after surprisingly long drawn-out negotiations on a trade treaty. Behind the published clauses of the treaty, there lies an attempt by Russia and the Stalinist Chinese leaders to change China's economy, and to exploit the strategically and industrial important areas of Manchuria and Sinkiang for the benefit of the Soviet Union. It is obviously Moscow's intention to make the strategic and economic development of Manchuria fit its own purposes. In Mongolia, but even more in Siberia, the Russians are developing a military position for which an industrialized Manchuria will be the main supply base.

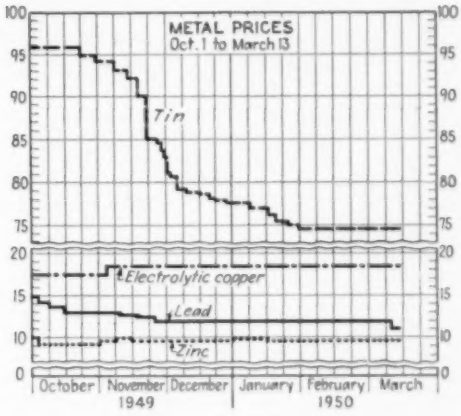
Already the links are close; estimates based on many reports from the spot put at 300,000 the number of Russian officials, and industrial and agricultural experts, who have displaced the half-a-million Japanese that were there in 1945. Most of Manchuria's trade is with the Soviet Union; its wheat, soya beans, meat and milk are exchanged for Soviet clothing, lumber and fuel oil. Private trade has shrunk almost to vanishing point as the Communist "Trade Trust" has expanded its operations. The plan is quite simple. Siberia milks Manchuria and Manchuria will milk China.

And, while Mao Tse-tung tries to explain all this to the Chinese, one of Stalin's own bully-boys has had to do some explaining on orthodox socialistic doctrine in a manner strange to outsiders. Mr. Andreyev, a member of the Politburo since 1931 holding a brief for agriculture, has been tartly criticized in "Pravda." He thought things over for ten days and in a letter to the editor promised to reverse the policy he has enforced for ten years and get along a little faster on farm collectives. This is an excellent example of how Stalin checks on his subordinates. Although the Politburo is so small and although Andreyev is intrusted with responsibility for agriculture, somebody has apparently been detailed to keep watch on him and his policies and to reprove him when necessary in "Pravda."

In Berlin the Russians are tightening down the screws again. Western German businessmen who have in the past few months concluded barter deals with firms in Eastern Germany have been told "off the record" that they must undertake not to let Western Berlin industries have any of the goods or raw materials delivered by the Eastern German firms. This Russian maneuver has as its object the killing-off of Western Berlin's economy. Western Berlin is living on continued injections of ERP and D-marks from the Federal budget (to the extent of $60 million a year and D-marks one billion), which the Russians doubtless envisage will stop by 1952 at the latest. Then the unemployed of the Western sectors of Berlin will, it is hoped in Moscow, be ready to submit to the Communists. There is not a doubt that the Communist German spokesmen were uttering a serious threat when they declared recently that the "democrats" of the city must carry out as an urgent task the eliminating of the "imperialist bridgehead" in Berlin.

And, down in France, as if they don't have bigger things to worry about, the Council of the Republic has had before it and rejected a Bill, passed by the National Assembly with a show of hands, that would give the Ministry of Health unlimited discretionary powers over any beverage "derived in whole or in part from plants or vegetable extracts." Coca-Cola is not mentioned by name, and the Minister would have to consult the Council of Hygiene and the National Academy of Medicine, but everyone knows what is intended.

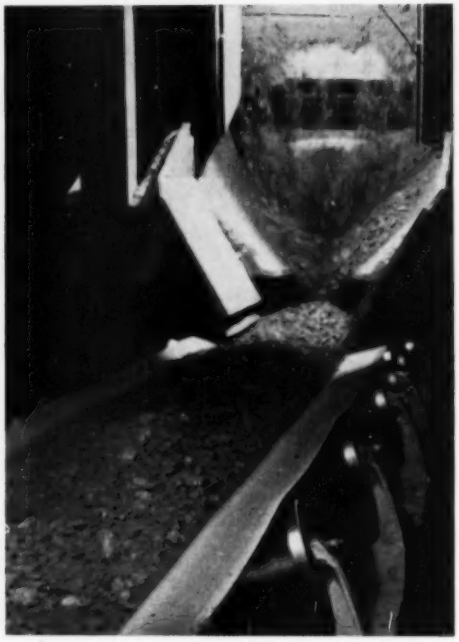
According to the London Economist, it is not easy to separate the streams of interest and indignation that have poured into the resistance movement against "coca-colonialism." The wine-growers in Hérault appear to have made the first protests in defense of wine; presumably it was not they who argued that Coca-Cola is injurious to health because they themselves produce local



wines whose effect on the liver cannot be entirely in the national interest. Then the Communists took up the defense of national health and the attack against "American imperialism" and soon had the press in full cry. Then the Prime Minister's own Progressive Republicans got involved and a Bill was sponsored by their M. Boulet from Montpellier. But M. Boulet soon found himself attacked from the flank by the Communists, who moved an amendment to his Bill naming Coca-Cola and banning it under the terms of the law. This, of course, had to be opposed, because the measure would then have constituted discrimination against an American product, which is ruled out by existing trade agreements.

So a new and not altogether minor action has begun in the cold war. The French having let off steam, and the Americans having had immense free publicity, they should call it a day. For any law that is passed can be evaded—for example, by adding rum—and any reprisals by the Americans will delight their enemies in France.

Beneficiation of Adirondack Magnetite



• • •

by W. R. Webb and R. G. Fleck

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Iron ore mining in the Adirondack region of northern New York dates back to the Revolutionary War. It is recorded that Benedict Arnold in his campaigns in the Lake Champlain area during the American Revolution utilized the local ores to produce armament and cannon balls. Through the efforts of early pioneers, the iron mining industry in northern New York gained impetus so that in the 1880's mines in the area accounted for nearly one fourth of the iron ore produced in the United States. Both magnetite and red hematite ores were mined during this period. With the discovery of the Lake Superior ores, the iron ore industry in New York started to decline and remained relatively unimportant until 1938.

Modern day methods of mining and beneficiating sufficiently high tonnages of low-grade ores to justify the large expenditures for plant and equipment prompted a resurgence in northern New York mining.

During the past several years, the production of iron ore from the Adirondack magnetites has greatly increased, and production which amounted to .6% of the United States total during the 1918-1928 period (*Mining and Metallurgy*, November 1943) has been increased to 2.85% of the total in 1948. Today, Republic Steel is mining in the Port Henry-Mineville and Lyon Mountain areas; the

M. A. Hanna Co. is mining at De Grasse; and the National Lead Co. is mining a high titania ore at Tahawus, approximately 100 miles east of Jones & Laughlin's Benson operation.

Published tonnages (*Mining Engineering*, March 1949) for the year 1948 show that the four Adirondack operations mined and processed about 7,230,000 G. T. of crude ore from which 2,868,000 G. T. of magnetic concentrates were produced. About 70% of the concentrate tonnage was sintered at the mines. In addition to the iron ore produced, the National Lead Co. extracted 250,000 G. T. of ilmenite from their magnetite-ilmenite ore. The operations which produced the 1948 tonnages are:

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Mine	Crude	Concentrate (Gross Tons)	Sinter
M. A. Hanna, De Grasse	200,000	128,000	128,000
National Lead, Tahawus	1,500,000	575,000	490,000
Republic Steel, Port Henry	1,868,000	969,000	663,800
Republic Steel, Lyon Mountain	1,273,575	386,100	
Jones & Laughlin, Benson Mines	2,390,363	809,587	726,058

These crude ore tonnages do not include the waste rock that was selectively mined and discarded before crushing.

The presently operated magnetite ore properties were all discovered more than 100 years ago and all, in fact, were mined at some time between 1824 and 1889. Of the current Adirondack mining developments, the Benson mines operation of Jones & Laughlin Ore Co. was the last to go into production. The beneficiation plant began to operate on a testing schedule in November 1943, using as ore the tailing from the previous operations which had produced iron ore intermittently between 1889 and 1918. Actual open-pit mining operations were begun in February 1944, and except for two labor dispute interruptions have been conducted on a year-around basis since that time.

The Adirondack magnetite deposits are generally similar in nature, and the beneficiation processes that are used to extract the concentrate differ only in the degree of grinding that is required to physically unlock the magnetite from the gangue. Principal gangue minerals are feldspars, quartz, and mica; and some pyrite, apatite and garnet are present. All of these magnetite deposits contain some TiO_2 as ilmenite associated with the magnetite, and in this respect the National Lead operation at Tahawus is unique in that the ore processed is sufficiently high in ilmenite to make possible the recovery of an ilmenite and a magnetite concentrate.

Mining

Since the Adirondack magnetite concentration operations are basically similar except for the National Lead double extraction process to recover both magnetite and ilmenite, a description of the Jones & Laughlin operation will be used to illustrate a typical eastern magnetite operation.

The Republic and M. A. Hanna operations are underground mining developments, while National Lead and the operation to be described are both open-pit developments.

The ore body operated by J&L is a long, comparatively narrow deposit. The length is slightly over two miles and the width varies from about 400 to 1400 ft. The ore body dips at about 60 degrees under a hanging wall of granitic gneiss. The iron content in a zone about 80 ft wide adjacent to the hanging wall runs about 32%

At right, the Republic Steel Corp's. ore separator in action at Lyon Mountain, New York. On the facing page: (top) crushed rock on the way to the separating plant at the J&L open-pit mine at Star Lake. (bottom) Titaniferous magnetite rock entering a primary ore crusher at the National Lead Co. operations, Tahawus, N. Y.

Two stages of wet belt type magnetic separators such as these are used at the J&L operation. After primary separation, the concentrate passes over three vibrating screens and then is returned to the secondary separators, where a concentrate averaging 63.75% iron is produced.

magnetic iron which gradually decreases in the direction of the footwall. The footwall, then, is an arbitrary grade limit so established that the over-all magnetic iron content of the mineable ore body is about 24%.

In addition to the hanging wall waste rock which must be removed to uncover the main ore zone, it is necessary to remove surface which varies from nothing to about 60 ft in depth and consists mainly of glacial drift—sand and boulders. Surface stripping is accomplished by shovel, drag line or dozer, or a combination of the three depending upon the nature of the terrain and the depth of material to be removed.

The ore to be quarried is a hard rock formation requiring controlled drilling and blasting practice to compensate for variations in nature of breakage of the several operating faces.

Nine-in. primary drill holes are put down with 42-T churn drills. Hole spacings average 26 ft and burden at the toe of the face is 20 ft. Nitramon and nitramex, in 9 in. and 6 in. cans, are used to distribute the explosives column and eliminate decking. Plant tailing is used for stemming. The loads are detonated with primacord which is fired by a time delay electric switch using an interval of .020 seconds between holes. Four tenths of a pound of powder is required per long ton of crude ore broken in the primary shot.

All pieces larger than about a 4-ft cube are block-holed with an air hammer and tungsten-carbide bit and blasted with stick dynamite before shovel loading. The secondary blasting operation requires .016 lb of powder per ton of original crude ore broken. Hence, a total of .416 lb of powder is consumed per long ton of crude ore broken to crusher feed size. Marion 4161 electric shovels equipped with 4½-yd dippers are used to load the ore into 22-ton end dump Euclid trucks for transportation to the coarse crushing plant. Present hauls are level or downgrade and average length of haul is about one mile.

Crushing The coarse crushing plant consists of a 54-in. Allis-Chalmers gyratory primary crusher and two Allis-Chalmers 18-in. secondary crushers which reduce the run-of-mine rock to 7 to 9 in. and 2¼ to 4 in., respectively, depending upon the amount of wear on the crushers. The secondary crusher product is belt-conveyed to surge bins which feed the fine crushing plant.

The fine crushing plant utilized two 7-ft cones as third stage crushers and one 7-ft cone as a fourth stage crusher. The product, essentially all minus ⅝ in., is belt-conveyed to three storage silos which feed the concentrating plant.

Concentration The concentration of a low-grade magnetite ore involves grinding to a size which will physically unlock the iron oxide particles from the waste gangue, and subsequently removing the iron oxide from the ground mixture by a magnetic separation.

The cost of grinding and the difficulties encountered in subsequent filtering and sintering operations on very fine concentrate must be considered along with iron recovery when determining the size to which the ore is to be ground. On our Benson ore, magnetic iron recoveries of 95 to 96% are obtained by grinding through 20 mesh and producing a concentrate which will contain an average of 45% minus 100 mesh material. A typical sieve analysis of finished concentrate shows:

Mesh	Percent	Cumulative Percentage
+ 20	.4	.4
+ 28	3.8	4.2
+ 35	10.0	14.2
+ 48	12.6	26.8
+ 65	13.0	39.8
+100	14.4	54.2
−100	45.8	100.0

The concentrator is three-section plant, each unit containing a 9½ by 12-ft Allis-Chalmers rod mill, a bucket elevator, three 48-in. wet-belt type primary electromagnetic separators, three 4 by 5-ft hummer screens with 20 mesh cloth, three 48-in. wet-belt type secondary electrimagnetic separators, and an 8 by 8-ft Allis-Chalmers ball mill and 54-in. Akins classifier in closed circuit for regrinding, a 9 by 14-ft Dorrco drum-type filter, and the necessary pumps and launders to transport the products between the units.

The tailing from three sections is collected in a common launder and carried to the disposal pumps.

The crude ore, after four stages of crushing, is fed to the rod mill at a controlled rate of from 100 to 135 long tons per mill hour. The rod mill discharge contains 80% solids ground to a size which distributes 15% on a 20 mesh and 28% through a 100 mesh screen. This is an open circuit operation, and the entire rod mill product is lifted by bucket elevator to a rotating pulp distributor located on the uppermost floor in the building.

The ground, unsized pulp is distributed to the 48-in. wet-belt type primary magnetic separators operated on 240 volt d-c current. Since the concentration ratio is close to 3 to 1, the primary separators are operated, without the usual variable resistance, on full current to pick up essentially all of the magnetic iron and drop a clean tailing. In this manner approximately 45% of the original crude ore is discarded as waste on the first stage of separation. The magnetic iron in the tailing for the Benson operation to date has averaged 1.93%.

The concentrate product from the primary separators contains considerable locked material and will average about 52% iron. This rough concentrate is screened over three 20-mesh vibrating screens, and the oversize is laundered to a ball mill, the undersize going direct to three second stage 48-in. magnetic separators on which the intensity is controlled by rheostat to produce a selective concentrate which to date has averaged 63.75% iron. The locked particles which were picked up by the primary separators and dropped by the secondaries constitute a middling which is transported to the ball mill classifier circuit for regrinding along with the plus 20 mesh rough concentrate from the screening operation.

The ball mill is operated in closed circuit with a classifier, and the classifier overflow which contains about 20% solids and a screen analysis of essentially all minus 35 mesh is pumped to the pulp distributor and reenters the separator section along with the new feed from the rod mill. The recirculated load from the regrind circuit constitutes about 40% of the total feed to the primary separators.

The finished concentrate from the three second-stage separators in each section is laundered to a settling cone which permits decanting a portion of the water, and the cone underflow containing about 50% solids is piped to a 9 by 14-ft Dorrco drum filter. The filtered concentrate containing 8% moisture is discharged onto a conveyor belt for transportation to the sinter plant, to railroad cars for direct shipment, or to a stock pile when cold weather prohibits the shipment of this moist concentrate.

The most satisfactory filter media for this type of operation has been a 1100-opening-per-square-inch rubber blanket called "Multipore," the manufacture of which was discontinued several years ago. Unless a satisfactory filter media substitute is developed, it is expected that much of the present satisfactory filter efficiency will be lost when existing supplies of the rubber media are exhausted.

The combined tailing from all primary separators are laundered to one of two 70-ft thickeners which can be used to reclaim water during a period of shortage, but normally serve only as a large-capacity sump to accommodate a variable tonnage of tailing and feed two rubber-lined sand pumps which operate in series to pump 5000 to 6000 tons of tailing per day.

The material pumped contains about 25% solids by weight and is currently raised about 70 ft and carried from 2000 to 4000 ft, depending upon the portion of the tailing basin which is being filled at the time. Tailing is impounded in an area of 378 acres by a berm constructed from mine stripping and/or the coarse portion of the tailing. The associated water is decanted from the basin and tiled from the area to find its way back to the small river from which it was originally taken. Approximately 5 million gallons of water are used daily. The normal stream flow is supplemented by a 380 million-gallon reservoir to insure a steady supply.

Despite the fact that some blending is accomplished by mining from two or more faces simultaneously, rather wide variations in iron content and physical character of the crude ore are noticed in the concentrating operations. While the over-all concentration ratio, that is, tons of crude ore per ton of concentrate produced will run just under 3 to 1, individual daily performance has been known to vary from 2 to 4 to 1. Grindability as determined by standard laboratory rod mill tests have shown that the weight ground to minus 14 mesh per mill revolution varies from 20 to 40 grams or a variation of 100%.

The variations in iron content of the crude ore will naturally be reflected in the relative tonnages of concentrate and tailing produced. It is, therefore, necessary to have some reserve filtering and tailing disposal capacity in order to accommodate the variable tonnages handled by these units. In extreme cases it is necessary to adjust rod mill tonnage in order to accommodate the unit which is being overloaded at the time.

Variations in grindability affect the size of product produced in the rod mill operation and hence the quantity of material finding its way into the regrind circuit. Insofar as is possible, the grinding media load is established to give satisfactory results on the type of ore being currently processed; but unforeseen changes in the physical character of the ore require an adjustment in the rod mill feed rate to balance the ball mill regrind load.

The rod mill discharge is sampled by an automatic sampler every 20 minutes and is prepared and analyzed to maintain a check on the crude ore analysis. The tailing is likewise sampled automatically and analyzed. The finished concentrate is checked every 4 hr for moisture, and 4-hr samples are sieve-tested for sinter plant control. Eight and 24-hr composites are analyzed, and ratios and recoveries are calculated for plant process control. A typical analysis of the crude ore and concentrate shows:

	Soluble Iron	Magnetic Iron	Phosphorus	Sulphur	TiO_2	Manganese	SiO_2
Crude	26.75	24.00	.192	.48	.68	.25	40.80
Concentrate ..	63.75	62.54	.024	.21	.77	.25	5.66

Up to this point we have discussed the Benson ore body as a magnetite deposit. Actually the ore mined to date has contained 2.94% of nonmagnetic iron in addition to the magnetic iron content. While the separation process is intended to

concentrate the magnetic oxide, the close association of the nonmagnetic ore present permits the accidental recovery of some of this nonmagnetic ore along with the magnetite, as indicated by a difference of about 1.20% between the magnetic and soluable iron contents of the finished concentrate.

The ore body is variable in its type of iron content, running from nearly all magnetite in some sections to nearly all nonmagnetic in others. The various grade zones are, of course, well determined by drilling; and the present mining operation is confined to those portions of the ore body which are low in nonmagnetic iron content.

Considerable pilot plant research has been conducted to develop a process to concentrate those ores which contain an appreciable amount of their iron content in the nonmagnetic form. The Humphrey spiral concentrator has been found to do a satisfactory separation job on these mixed ores, and the results have been confirmed by processing 50,000 tons of crude ore through a small experimental spiral plant.

In order to fully exploit the ore body, it will be necessary to construct a production scale gravity separation plant to handle the ores which contain sufficient nonmagnetic iron to make magnetite separation impractical.

Since the expected over-all recovery in a gravity plant will be considerably lower than the magnetic recovery in a magnetic operation, it remains to be determined at what composition the dividing line will be established between magnetic plant and gravity plant feed.

The gravity plant concentrate will be higher in phosphorus and titania content than our present concentrate and will be somewhat lower in iron content. This reduced ore value is, of course, a second factor in determining the nonmagnetic iron content that will be established as the dividing limit for the two types of treatment.

The concentrate produced from the Adirondack magnetites is, of course, too fine for direct use in a blast furnace and must be agglomerated before it can be considered a blast furnace ore.

The final step—iron sinter being loaded into a railroad car, enroute to blast furnaces.

The four operations in the area use sintering—three, including the Jones & Laughlin operation, employing the Dwight Lloyd continuous-type machine and one operation using the Grenawalt batch-type process. Each operation uses fine anthracite coal, either No. 4 or No. 5 buckwheat size, for the mix fuel.

Conclusion

We have reviewed briefly the current activities in the Adirondack region. Now let us examine the potential future of the operations in this area.

It is entirely possible that future developments will not stop at the figures shown in the present production schedules. Economic or international conditions may make it necessary to increase the production rate. Would this mean a rapid depletion of the Adirondack reserves? Not necessarily. There are many possibilities in the Benson area; and, no doubt, in other parts of the Adirondack region.

Our geologists as yet do not have complete data on the underground ore, but it is safe to say that the portion of the ore body that could be removed by underground mining methods contains large additional tonnage above that now shown in the open-pit reserves.

In the vicinity of Benson mines there are a number of smaller magnetite deposits of varying richness. While these relatively smaller ore bodies could not be developed economically under present conditions, the time may not be too distant when they will provide iron ore for our blast furnaces.

Probably the greatest potential reserves at Benson are the vast deposits of iron-bearing rocks that are too low in iron content to be classified as ores under today's price and cost structure. The current estimates of reserves are based on the concentration ratio of 3 to 1 (3 tons of crude required to produce 1 ton of concentrate). If we included material with a 4, 5, or even 6 to 1 ratio, the figure on reserves would be increased many fold. But would it be economical to mine and beneficiate such low-grade material? The answer to this question: Yes, provided the operation was on a sufficiently large scale. The copper mines in the West are excellent examples of what can be done with very low-grade ores. Some of the copper ores being mined at present contain less mineral values than Benson iron ores with a 6 to 1 concentration ratio and are considerably more expensive to beneficiate. To obtain low unit costs, however, the copper companies must handle tremendous tonnages, with some mines daily removing more than ten times the amount of waste rock and ore being mined at Benson today. The expansion of Benson to an operation of this magnitude is entirely within the realm of possibility.

Acknowledgment

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The Critical Decades

AN eloquent plea for clear and objective thinking when, with a war behind us and the possibility of future conflicts looming large, we must re-evaluate our mineral position with an eye to defense.

by Harrison Schmitt

ARE we a "have" or a "have not" country in mineral resources? Like many debates that generate heat much of the trouble results from the lack of definition. We are unusually well off in many basic mineral resources, but apparently deficient or lacking in others. Progressively our position will be a function of the rate of depletion, of imports and consumption, of substitutes developed, and in emergency, that is, war, the time available for utilizing low-grade deposits. The meat of the question is whether we will be a "have" in critical materials in case of war in the near future, particularly if for defense we should be thrown back to the Western Hemisphere. This latter possibility is taken seriously by this writer who can't forget the Pearl Harbor fiasco, the 1944 break-through in France and other serious military blunders of the last and indeed of most our wars. I have little confidence, furthermore, that the past and present average United States diplomacy will protect our interest and status in the Eastern Hemisphere where we seem to be losing many of our hard-won outposts by default. I think the possibility that potential enemies might gain a sudden advantage by treachery or a new weapon must always be in mind.

The data given in **Mineral Resources of the United States** by the Bureau of Mines and the Geological Survey, published in 1948 (Public Affairs Press, Washington, D. C.) appear to be good basis for judging our position now and our position as it might become. Now, and for the next few years at least, we stand largely dependent on foreign sources for: chromite ore of the various types, ferro-grade manganese, nickel, platinum metals, tin, industrial diamonds, quartz crystal, spinning quality asbestos, cobalt, and flake graphite. We are partially dependent on foreign sources for petroleum, arsenic, bismuth, cadmium, copper, iron ore, lead, mercury, tantalum, tungsten, zinc, acid grade fluorspar, antimony, vanadium, high-grade bauxite and strategic mica.

This list will change with changes in technology, substitutes will be found, and in case of war low-grade, impure and refractory reserves will be used to some extent. Further, if we can hold most of North and South America our position will be good, particularly in antimony, arsenic, bauxite, bismuth, cadmium, copper, iron, lead, mica, nickel, petroleum, platinum, quartz crystal, tin, tungsten, vanadium, and zinc. Unless accumulated in stock piles, however, and these should be large enough for a five-year war, we will be in need of: asbestos, chromite, industrial diamonds, graphite, and manganese. For a five-year war we would need to have stock-piled around five million long tons of the various kinds of chromite ore and six million long tons of manganese ore. If we assume that *in extremis* we might be thrown back to the continental United States we should stock-pile in addition: bauxite ore, bismuth, cadmium, copper, iron ore, lead, mica, nickel, platinum, the platinum metals, quartz crystal, tantalum, tin, tungsten, vanadium and zinc.

This seems to be a measure of our "have-have not" position, as we now face it. Many of our great mining camps have been declining in production for several decades according to the statistical branch of the Bureau of Mines. We are finding and will continue to find new mines, doubtless even districts, by increased exploration rates, the use of new tools in engineering, geology, and geophysics and perhaps by accelerated

Federal aid. Improved technology will make ore out of low-grade material and productive mines out of prospects. The realistic view, however, and, at least, the precautionary national view, is that this will most likely merely decelerate the decline in the rate of production. One can't emphasize too much that we are mining wasting assets and, furthermore, that production demands are ever increasing.

In this day when pressure groups are the fashion—one must admit they are defensive in part —when most people are intent on getting "theirs" and the devil take the rest, one actually feels apologetic, even on the defensive, if one ventures to take the properly objective view on economic and other matters. Then, too, if one pretends to the scientific approach one is bound to try to look at these questions objectively. Even from the purely selfish view, moreover, it would seem unequivocal that what benefits the country as a whole will also benefit the individual. If you can accept this as valid we have a right to proceed.

Metals and minerals in adequate supply at all times to keep pace with our ever expanding needs and responsibilities in peace and for defense, are, it goes without saying, an absolute must. If we cannot get them at home we must get them abroad. The payment of subsidies, adequate to close or narrow the gap between production and consumption in peace time seems a weak approach to the problem from the view of the national interest. At best it parallels the treatment being given the consumers and taxpayers in other commodities. You are taxed for the doubtful privilege of paying higher prices. It now appears that if we do not reverse our policy in grains, potatoes, cotton, etc., we are faced with some sort of a national fiasco compounded of non-disposable surpluses, still higher taxes and/or budget deficits. This is against the national interest and on the long pull will have to be abandoned even if it is now firm policy for this nation to allow itself to be blackmailed by powerful if privileged groups. We all seem to be in favor of a free or even a depressed market, but only for the *other* fellow's goods. Note, for example, the case of rents.

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The majority of informed persons agree that in the history of this nation the next few decades will probably be among the most critical. For survival we must give more and more attention to defense preparation. Paramount of course is strong social unity and a powerful industrial plant, but when it comes to actually making the machines for food production and fighting, metals and minerals are now a top critical factor. Our past international blunders of state during and since the war, plus the fanaticism and malicious ill-will of one avowed enemy, have set up a process of encirclement which has not yet been stopped. This plus some bad luck can now conceivably isolate the North American continent in case of war. If you agree that we can no longer let national defense be a matter of chance, luck, and time for preparation granted by the grace of God you will perhaps be forced to support much of the following:

(1) A stock pile of metals and nonmetals must be built up adequate to take care of shortages that could conceivably develop in a long defensive war.*

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* According to the *U. S. News-World Report*, pages 51, 52, March 18, 1949: "Buying of manganese and chromite will be stepped up whenever those products—vital in steel making—are available. . . . It is probable that there will be accelerated purchasing of lead for the stockpile. . . . Many other metals are to be ordered—for immediate or delayed delivery. Copper, tin and zinc are accumulating in military depots all over the country. . . . On hand so far is $821,000,000 worth of strategic materials—22 per cent of the total that eventually is to be accumulated. . . . In the next 15 months the Munitions Board hopes to place orders for another $736,000,000 worth of materials. . . ."

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(2) Since once mined and consumed bodies of minerals are irreplaceable it is questionable whether, by tariffs and subsidies, we should deliberately encourage depletion of the ones that appear to be in short supply.

(3) Since a lower return on mining will close marginal mines and make it difficult to reopen them quickly, the government should perhaps by subsidy keep such mines on a stand-by basis.

(4) Within feasible political and economic limits we should not be reluctant to use foreign metals and minerals that will come to us in normal trade. Have we possibly come to the point where the national interest would be served if mines are paid for not mining the more critical minerals or, in any case, for freezing their production in stock piles and using imports for current consumption? Apparently serious consideration is being given to cease mining the Mesabi range iron ore reserves when sufficient foreign ore can be imported, according to recent news reports. Should we in the future export any of our strategic materials except to reliable and potential allies who need strengthening?

Since we sell abroad by means of foreign exchange that is created not by imports but by money gifts and by loans that are seldom repaid we are literally giving away many of our irreplaceable resources. The widely held theory that we must somehow dispose of our surplus wealth to insure prosperity at home is, of course, encouraged by foreign recipients of our bounty who at the same time are greatly amused by our economic naïveté. In conclusion, it seems likely that circumstances will force us, if we are to survive, to abandon barroom and cracker barrel ecomics and to think more in terms of the national interest and contemporary world history, all of which has a critical bearing on the mining economy.

The writer is greatly indebted to Mr. Frank Light of Silver City, N. M., for criticism of this paper.

Before the Legislatures of the
Federal Government and the Several States

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Docket No. 1
In Equity.

Taxation, *Plaintiff*

vs.

Mineral Resources, *Defendant*

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A SUIT FOR DELINEATION OF BOUNDARIES

Brief by GRANVILLE S. BORDEN, a Friend of the Court

This brief is a reconnaissance report on some prospects in the field of taxation which can safeguard our Mineral Resources. The brief is a prospectus soliciting help and enthusiastic cooperation in the common task of developing these prospects in a manner which will be most beneficial to all parties—the owners, operators, employees, tradesmen, the federal and state governments, and the people of this nation.

IN the Constitution of the United States and in the Constitution of the several states, the people have delegated to their legislators the powers to levy taxes upon them. Notwithstanding certain constitutional limitations upon these powers, such as due process clauses and equal protection clauses, the power to tax is the power to destroy. These legislators, in whom the power to tax is vested, can aggravate inflation; they can cause depressions; they regulate our standards of living; they can destroy free enterprise; they can tax special groups and give the proceeds to other groups or foreigners; they can levy excises which will regulate any type of activity; they can, by devious means, confiscate and expropriate; they can, through discriminatory taxes, engender civil strife; they can make us vulnerable to foreign aggression; and last but not least, as shown herein, they can convert our Mineral Resources into waste.

Although few of our legislators have had experience in the business of finding, developing, and producing our fuels and basic raw materials from the earth, the presumption seems sound that they are curious and interested in these activities; that they welcome instruction, especially from informed citizens when given in good faith and with the intent to promote justice and equality for all our people.

In the light of these circumstances, what can we do? We should strive to instruct our legislators in the impact of taxation upon our Mineral Resources to the end that they, through their votes, can construct barriers inside our Mineral Resources to prevent irreparable damage from trespassing by taxation. We can as individual citizens offer instructions and training to our tax lawmakers.

There are many facts which every lawmaker should know as he surveys the traverse of taxation through our Mineral Resources. Some are of broad, general import; others relate to current, specific problems. Some seem elementary; others are quite complex and technical.

As *amici curiae* (friends of this court) some of these facts are recorded herein where they may be available for assignment to our tax lawmakers. The balance of this brief is a recital of some of these facts.

What Every Tax Lawmaker Should Know

1—The magnitude of our Mineral Resources varies inversely with the magnitude of the taxes levied upon them. Mineral Resources are natural deposits containing valuable ingredients which can be exploited profitably. When the proceeds from the disposition of the valuable ingredients of any deposit are inadequate to meet the costs of production, the deposit loses its label as a Mineral Resource. It is converted into waste. As taxes on Mineral Resources are increased, more marginal deposits are converted into waste. For example, if the recoverable grade of ore is $10 per ton and the production cost is $9.50 per ton, this ore becomes waste if the state imposes a severance tax of $1 per ton.

2—Many additional levies on our Mineral Resources have effected net losses in revenue to the government which imposed them. Our Mineral Resources are a substantial portion of our national wealth and a major tributary to the stream of our national income. The proceeds from the disposition of the valuable ingredients are distributed to government, as taxes; to labor, as wages; to capital, as profits and dividends. As labor receives its wages, the income is taxed; as capital receives its profits and dividends, the profits and dividends are taxed. If the profits are corporate profits there is a double tax—on the corporation and on the dividends distributed by the corporation.

As the tax rates increase, Mineral Resources decline. When the burden becomes too heavy, mines and quarries are shut down, ghosts invade our mining camps, men lose their jobs, suppliers lose their trade, stockholders lose their dividends, processors and refiners lose their raw material, and Uncle Sam and the state lose their taxes on the profits of the operator, the wages of labor, the royalties of the owner, the profits of the tradesmen. Thus, although the tax rates have been increased, there is a net decline in the flow of revenue.

Not only does the loss of revenue ensue from the reduction in yield of income tax, but every increase in tax on the Mineral Resources effects some reduction in the fair market value of the deposit. With this decline in value, there are proportional declines in the yield of: (a) the *ad valorem* tax of the state, (b) the state's inheritance and gift taxes, and (c) the federal estate and gift taxes. Thus, although the tax rates have been increased, there frequently is a net decline in the flow of revenue.

3—Tax increases on our Mineral Resources reduce the number of discoveries of new Mineral Resources. Exploration is an expensive, speculative activity. Every tax increase tends to evaporate the funds available for exploration, and every tax increase tends to impair the incentives to enter or continue in this speculative and expensive exploration business.

If our standards of living are to be preserved, discoveries of new mineral resources are imperative: First, to set off the current depletion of known resources; secondly, to provide adequate reserves for growth in population under an expanding economy; and finally, as a stock pile to serve as a potential supply in time of war when

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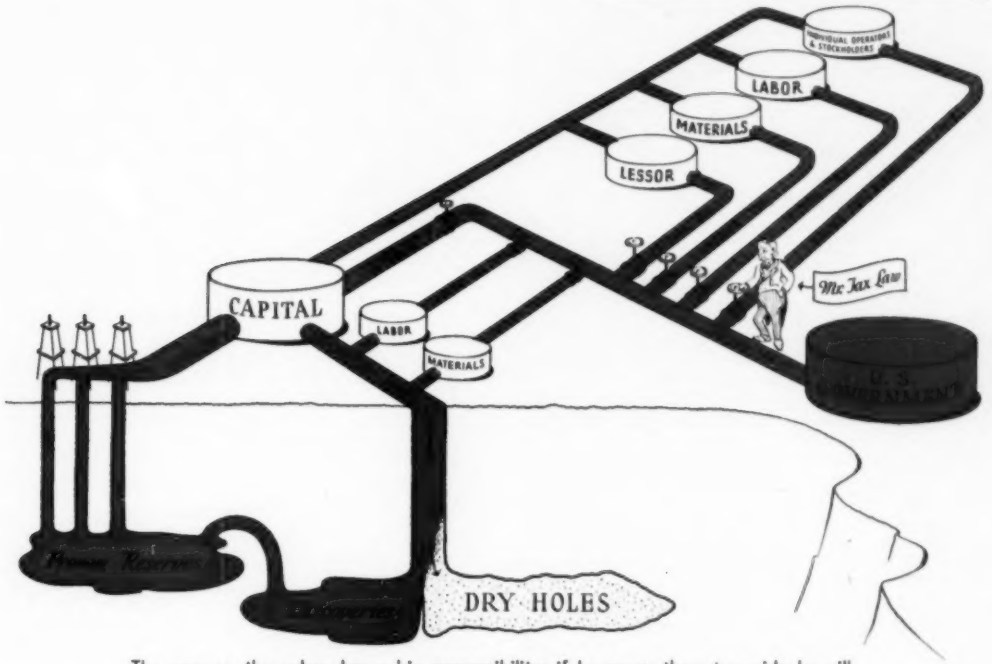
Mr. Borden is a mining engineer, lawyer, and AIME Member from San Francisco, Calif. **This paper was presented at a Mineral Economics Division session during the AIME Annual Meeting, February 1950. The opinions expressed are those of the author and not necessarily of the AIME.**



The man on the valves has a big responsibility; if he opens them too wide he will empty the pipe line faster than fresh supplies are discovered.

there are extraordinary demands coupled with man-power shortages.

No one can forecast how or when nature will yield the secrets of her underground deposits. Discoveries are not made under a time table, and the quantity and grade of the new deposits cannot be forecast by the best experts in the geological or economic sciences. All we know is the fact that the probabilities of discoveries depend on the amount of exploratory expenditures and the enthusiasm and wisdom of those who do the hunting. But no one spends money to hunt for worthless game emaciated from a superfluous tax diet.

4—Every discovery of a new Mineral Resource creates new sources of revenue for the federal and state governments, new sources of employment, new stimulants to trade, new sources of funds for more exploratory ventures, and new barriers against poverty and foreign aggression. We hear of "incentives" for the mineral industry. Why not speak of "grubstakes" for those who take risks to create new national wealth and national income? From time immemorial sovereigns have granted awards for outstanding services. The prospectors who create new national wealth should be the object of some governmental bounty.

Every discovery is made at substantial risk of loss. These risks increase in inflationary times as the quality and quantity of the prospects diminish.

If the government should feel justified in providing some grubstake for those who agree to explore, such action should not be deemed to be discriminatory against other taxpayers. There are many types of class legislation which extend special benefits in the name of the public interest—loans of the Reconstruction Finance Corp., farm subsidies, cooperative exemptions, the exemption from income taxes of all the income of labor unions, charities, educational institutions.

5—As additional taxes dry up the source of taxes, more taxes are imposed to meet the deficit; thus the merry-go-round of depression is accelerated.

Income Taxes

6—Every tax lawmaker should know that the producers of minerals, oil, and gas have not had a fair deal in the evolution of the federal income tax law; that the income tax burden levied upon them is, in many instances, discriminatory and unjust.—This discrimination arises primarily from the failure of Congress to recognize that a producer of minerals is in substance and reality engaged in two business enterprises. First, he is selling his mineral deposit. Secondly, he is in the business of mining. The over-all profits from both businesses are taxed as ordinary income at the normal and surtax rates; but if he sold the deposit to you and you mined the deposit, he would be taxed at the long-term capital gain rates on the difference between the sales price (value in place) and the cost.

We can concede for the purpose of discussion that the profits derived from the second business —i.e., the mining activity—should be taxed as

ordinary income, but the profit derived from the sale of the deposit should not be taxed more than profit derived from the sale of any capital assets.

To illustrate, in the case of a corporation, except relatively small corporations, the tax rate is 38%. Assume it paid $10,000 for some property and discovers a deposit worth $510,000. If this corporation sells the deposit to you, its federal income tax is $125,000 (25% of $500,000). If the corporation realizes this appreciation of $500,000 by mining the ore and selling this ore on the surface, it will pay on this realized appreciation $190,000 (38% of $500,000) less any reduction which may flow from the benefits of percentage depletion, which will be discussed next. Consider an open-pit mine. Why should the tax rate be 38% if corporation A, the owner, uses its own shovels and employees and only 25% if corporation A sells to corporation B, and the deposit is mined with the shovels and employees of corporation B?

Thus, if we grant that a producer is really selling his deposit as he mines the deposit, the conclusion seems clear that he should pay only long-term capital gain rates on this portion of his income. Congress by its failure to recognize this situation is effectively discriminating against our mineral producers.

It is noteworthy that Congress has granted taxpayers who cut timber the privilege of having their profits taxed at the rates applied to gains from the sale of capital assets.

7—Every tax lawmaker should know the facts which justify the allowances of depletion deduction on percentages of income.—On several occasions in the past, Congress has considered suggestions from the Treasury to repeal the provisions permitting determinations of depletion deductions on percentages of income from the depletable properties. On each occasion, after comprehensive hearings before Congressional committees, the Treasury's proposals were rejected.

In President Truman's tax message to Congress on Jan. 23, 1950, he stated that there is no loophole so inequitable as excessive depletion exemptions enjoyed by oil and mining interests.

This is a serious charge against our Mineral Resources. If it is true that there is an inequitable percentage depletion loophole in the law, it should be closed. If there is none—if these depletion allowances can be justified—then this Congress should follow the pattern of its predecessors when they rejected similar proposals.

The word "loophole" generally connotes some obscure, devious device of avoiding taxes, but the right to measure depletion by a percentage of income has been prescribed by statute for 25 years.

Percentage depletion is a complex subject, but some of the principal reasons for the allowance are briefly stated: Generally, percentage depletion fails to set off the discrimination against producers discussed in point 6 above. As a rule, if *A* mines his deposit, he will pay more income taxes, *even with percentage depletion allowances*, than *A* and *B* would collectively pay if *A* should sell to *B* and *B* should mine the deposit—a case where no percentage depletion would be allowed.

This gesture by Congress to equalize the burden and remove discrimination cannot fairly be lodged in the category of a loophole.

Statistics will prove that the tax savings which flow from percentage depletion are generally added to other funds and appropriated to determine where nature has hidden other deposits. These hunting expeditions are expensive and all the ventures speculative. The cost of finding new reserves is higher than the cost of finding the reserves which are being depleted. This is true not only because of general inflation, but also because of the gradual decline in the quality and quantity of prospects. The merchant can replenish his inventory by purchasing new goods, but the miner replenishes his underground inventory by speculating and wildcatting. For example, to maintain our present domestic crude oil reserves, the prospectors and explorers must find 2 billion additional barrels annually, and this task requires a lot of fortitude whether you are speculating with your own funds or the funds of your stockholders.

Many of the important discoveries of the past would not have been made had there been no percentage depletion. In the first place, the funds in reserve available for prospecting would not have been adequate to support the more speculative type of ventures. In the second place, without some tax incentives the long-shot plays would never have been taken.

Even admitting that the government is in the grubstaking business, a review of the past will prove that the producers have repaid these percentage depletion grubstakes with thousands of percent interest. **The loss of revenue to Uncle Sam from percentage depletion is in reality a capital investment to serve all the people through the promotion of their security and their prosperity.** The net results of the percentage depletion allowances are stupendous gains to the government. The yield to our government from new oil fields—such as Cuyama in California, and North Snyder in Texas—will set off all the revenue losses from percentage depletion for a substantial period. From the standpoint of insurance for prosperity, percentage depletion outranks any subsidies, loans, exemptions, aids or benefits which are granted by our government.

Corporate taxpayers lead the exploration parade. Granted that percentage depletion gives these corporate producers some tax savings, how do corporations utilize the funds that are saved? As stated above, these tax savings are reappropriated to search for new deposits. These expenditures, even though fruitless, engender new sources of employment and new stimulants to trade. If a discovery is made, the government shares in the benefits. Assuming, although it is not a fair assumption, that all of these tax savings are not appropriated for the search of new reserves, what is done with the balance? This balance is paid to stockholders as additional dividends where Uncle Sam takes back his share on this increment as normal taxes and surtaxes at the top bracket of the shareholder. The tax saving of the corporate taxpayer may be lost down a shaft, winze, or a drill hole with the proportional loss to our government, but it is never poured down a rathole.

Finally, have our people forgotten the contributions of the mining prospectors and oil and gas wildcatters toward winning World War II? Had there been no long-shot wildcat discoveries in the prewar days of World War II of oil, gas, copper,

lead, zinc, and strategic metals, we would be paying for a much longer war. Do they remember OPA; the "A," "B," "R," and "T," gasoline rationing tickets? Do they remember allocating metals and the hysterical search for strategic minerals? If their memories fail, let them refer to our current stock-piling programs.

After this narrow escape from disaster (wars cannot be won without adequate domestic Mineral Resources), can we afford to smother the enthusiasm of our prospectors and wildcatters to the end that our underground stock piles will become exhausted from ordinary peacetime consumption so that we will be caught short of these indispensable sinews of war when the next sneak attack occurs?

As shown in the prior discussion of general principles, the inevitable result of a heavier tax load on the mineral producers will be a net loss in national wealth and national income, an impairment of our security from poverty and our power to defend ourselves against foreign aggression.

If the parties who insist that percentage depletion is a loophole should review the record carefully, they will undoubtedly reach the inevitable conclusion that any revenue which flows out of this loophole returns to our government with a flood-tide of accessions.

8—Every tax lawmaker should know that the rules governing exploration and development expenditures by mining, oil, and gas companies effect inequitable and unjust results. He should know the technical reasons for these unfair results. He should know how remedies can be procured.—Why does tax liability vary in the treatment of exploration and development expenditures? Because these expenditures may not be deductions from gross income in the determination of taxable net income: (a) if the expenditure must be taken as an expense in a year when there is not sufficient gross income to absorb the income (or used as a carry-back or carry-over deduction), or (b) if the expenditure must be capitalized to be recovered through depletion deductions over years when the depletion deductions are based on a percentage of income.

There is a broad disparity between the rules which apply to mines and those which apply to oil and gas.

The general rule for mines in the development stage is set forth in Bureau Regulations 111 (Section 29.23 (m)-15 (a)):

> "All expenditures in excess of net receipts from minerals sold shall be charged to capital account recoverable through depletion while the mine is in the development stage. The mine will be considered to have passed from a development to a producing status when the major portion of the mineral production is obtained from workings other than those opened for the purpose of development, or when the principal activity of the mine becomes the production of developed ore

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High on the eastern slope of the Sierras is the tungsten mill of the U. S. Vanadium Corp., one of the few producers of this metal of economic and military importance. The law should provide for writing off exploration expenditures so that more sources of tungsten and other strategic minerals may be found.

Courtesy of Curtis Phillips

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Courtesy Standard Oil Co. (N. J.)

Few industries face such technical difficulties as are encountered in drilling for oil in the Gulf of Mexico. Here the Humble Oil Co. is drilling seven miles off shore. The rig has quarters, store rooms, storage tanks and maintenance facilities.

rather than the development of additional ores for mining."

The general rule for mines in the producing status is that development shall be treated as deferred development expense to be extinguished at a rate commensurate with the exhaustion of ore it serves. *General Counsel's Memorandum 13954*, C.B. 1934, page 66.

In the case of oil and gas there is no distinction between development status and producing status.

All expenditures "for wages, fuel, hauling, supplies, etc., incident to and necessary for the drilling of wells and the preparation of wells, for the production of oil and gas, may at the option of the taxpayer be deducted from gross income as an expense or charged to capital account." But any election is binding upon the operator for all subsequent years.

The rules pertaining to the treatment of expenditures for geological and geophysical expenditures by oil and gas taxpayers are extremely complicated. The treatment depends upon a mind-reading test whether land or leases were acquired or retained as a consequence of the geological and geophysical expenditure. If this is the conclusion, the expenditure is capitalized; if not, the expenditure can be expensed.

Why is all this confusion necessary? Why should these rules be invoked to deprive the producers of tax benefits? Clearly, every taxpayer should be entitled to deduct the money which he has spent for exploration and development either as a loss if no discovery is made, or as deferred expenses of exploiting the mines which have been discovered and developed. The law should provide rules which will grant a tax benefit for every cent expended for exploration and development and prohibit the impairment of these benefits through (1) the obligation to take deductions in loss years, or (2) through obligation to take the deductions as cost depletion so as to be merged into percentage depletion deductions.

Incentives can be engendered by removing unfair "decentives." If the Treasury is unwilling or unable (due to provisions in existing law) to rescind these decentives, our Congress should act.

9—Every tax lawmaker should know that the mineral producer is particularly vulnerable to hardships from an unjust rule relating to the recovery of depreciable capital established by the Supreme Court of the United States. He should know the remedy.—Under a rule established by

the Supreme Court of the United States, taxpayers may be deprived of the tax benefits which generally flow from the right to take depreciation deductions based upon the capitalized costs of the depreciable assets.

Briefly, under this rule, if losses are sustained in any year (or the net income before depreciation is less than the depreciation allowable), the taxpayer loses a portion of the tax benefits to which he is justly entitled under the general rule that the wear and tear on depreciable assets used to produce income are deductible in determining taxable net income.

The rule, commonly termed the Virginia Hotel rule, is an exceptional hardship on the producers of minerals, oil, and gas because this class of taxpayer generally loses his rights to procure deductions for net loss carry-backs of carry-forwards through obscure technical limitations in Section 122 (b), (c), and (d) of the Internal Revenue Code, which will be discussed below.

Many taxpayers are advocating a just rule which will permit taxpayers to deduct the cost of their depreciable assets from gross income of any taxable years—that is, the taxpayer shall decide how much of the remaining depreciable cost shall be taken in any taxable year until the total cost has been recovered. This rule would be fair and eliminate all the complexities involved in current practice, and cure the sting of the Virginia Hotel rule. Can this decentive be eliminated?

10—Every tax lawmaker should know that there is an obscure provision in Section 122 of the Internal Revenue Code dealing with carry-back and carry-forward net loss deductions which has unjustly cost the producers millions of dollars and will continue to levy, indirectly, a terrific unjust burden on our Mineral Resources.—The discourse is addressed to an obscure, unjust technical provision in the Internal Revenue Code which has cost the mineral industry millions of dollars and will continue to levy indirectly an unjust tax burden on their deposits. There is one gold mining company in California where these provisions increased their taxes by $134,000 for 1943, and there is another California company operating a tungsten-antimony-gold mine in Idaho where these provisions increased their taxes for 1943 and 1944 by $200,000.

The provisions to which reference is made are Section 23 (s) and Section 112 (a), (b), (c), and (d) of the Internal Revenue Code. Hidden obscurely away in these confusing and bewildering sentences is the rule that the *net loss* of any year when carried backward or carried forward to serve as a deduction from gross income of another year, must be reduced by the percentage depletion allowances in excess of cost depletion for the year to which the net loss is carried. The adverse effect of this rule can be expressed in either one of two ways:

(a) It deprives the mineral, oil, and gas producers of the major share of the benefits which

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It is not the technical problems or rugged aspects of the business which discourage the mine operator; it is the tax decentives which actually make it expensive to explore and develop over and above operating costs.

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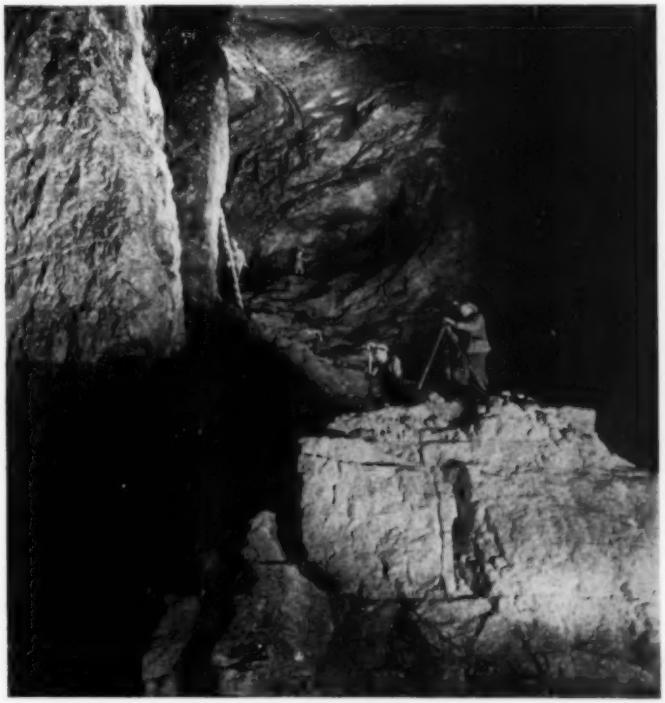


should accrue from the right to deduct net losses from the income of other years; or

(b) it indirectly repeals the right to a substantial share of the percentage depletion deduction for the year to which the net losses are carried.

The mineral producer falls in the category of feast and famine taxpayers who are vulnerable to wide fluctuations of income and losses over a series of taxable years. No other group of taxpayers are more equitably entitled to net loss carry-over and carry-back deductions. Nevertheless, these taxpayers, because of these obscure provisions in Section 122 of the Internal Revenue Code, can seldom procure any tax benefits from these provisions. The gold producers who sustained losses from War Production Board Order L-208 were the first group to lose refunds of prior year taxes because the losses sustained while operations were shut down were absorbed by the percentage depletion allowances of the year to which the carry-back loss would have applied. Next on the list of victims were the operators who mined stratetgic minerals at a profit during the war followed by an era of heavy losses in the postwar period.

Noteworthy is the point that the relatively small producers are the taxpayers most vulnerable to this inequity. Can this decentive be eliminated?

11—Income taxes can be assessed on the proceeds of new ventures before the operator has assurances that his investment will be redeemed.—The income tax law contains a provision in the section which allows deductions for depletion that "In any case in which it is ascertained as a result of operations or of development work that the recoverable units are greater or less than the prior estimate thereof, then such prior estimate (but not the basis for depletion) shall be revised and the allowance under this subsection *for subsequent taxable years* shall be based upon such revised estimate." (Italics supplied.)

If reserves prove to be less in a subsequent year than previously estimated, the cost depletion allowances for the year of overestimate may not be adequate to absorb the net income computed without the depletion deduction. Then, the operator pays an income tax. In later years, when as a result of "operations or of development work" the prior overestimate of reserves is demonstrated, the error cannot be corrected retroactively. The revised estimate of lower reserves can be used in determining depletion allowances and income taxes for "subsequent years" but no refund (except where carry-back losses may be deducted) can be procured of the taxes determined by using the erroneous high estimate. If the reserves as estimated prove to be inadequate to provide redemption, the taxpayer is out some income taxes even though the venture has not provided redemption of the amounts invested. Can this decentive be eliminated?

12—An investor in the shares of a corporation engaged in producing minerals, oil, or gas, may be assessed income taxes on dividends even though the amounts received are in fact redemption of the cost of his shares.—When a corporation distributes money or property to its shareholders, the test whether the distribution is taxable as a dividend (i.e., as ordinary income) or whether the distribution is redemption of the shareholder's cost of the shares is made from the financial status of the corporation; i.e., whether the corporation has undistributed earnings or profits, etc. The test is made without references to the cost of the shares to the stockholder. Thus, in many cases the shareholder must pay an income tax on all distributions which he receives from a corporate producer of minerals, oil, or gas, until the corporate property is fully depleted and liquidated.

For example, take the case of a purchase of stock in a mining company which is profitably mining an ore body—one which has small invested capital but substantial earnings from a limited ore body. All distributions are dividends and taxable income to the shareholders regardless of their investment in the shares. This is a rule of law, although in substance a portion of every distribution should be considered as redemption of the shareholder's cost of the shares to the end that the stockholder will procure complete redemption at the end of the life of the ore body and pay income taxes only on the increment of the distribution above the redemption requirements. Under existing law, the individual shareholder will pay income taxes on all the dividends at individual normal and surtax rates during the life of the enterprise, and upon its liquidation and termination he will sustain a long-term capital loss subject to the limitations on the amount deductible. Can this decentive be eliminated?

Conclusions

In conclusion, it is submitted:

1. The health and vitality of our Mineral Resources are in jeopardy from trespassing by taxes.

2. The duty and power to delineate boundaries and construct barriers against this invasion has been delegated to our tax lawmakers.

3. There are both elementary and complex facts pertaining to the taxation of minerals which every designer of mineral tax law should know.

4. Every citizen owes a duty to his country to help in this chore of teaching our tax lawmakers facts about our Mineral Resources to the end that they shall design and construct our tax laws intelligently—laws which will engender benefits for capital, labor, consumers, our state governments, our federal government, and every citizen of the United States.

Castle Dome Operating Ideas

by J. C. Van de Water

IN June the supply of M3 army demolition powder being used for mudcapping was depleted and we switched to 4 by 8, 60% Special Quarry Gelatin dynamite. This size cartridge has proved convenient, as a half stick will handle the average boulder. By keeping the gelatin in the paper while cutting it in two, there is little contact with the gelatin and therefore fewer headaches among the powdermen. To further help prevent the headache nuisance the men are furnished with plastic coated gloves.

Boulder breaking costs as reported last year were $0.426 using M3. This year, 8180 boulders have been broken with M3 at a cost of $0.472 each and 7479 with 60% Quarry Gelatin at a cost of $0.660 each. This rise in cost is due entirely to the greater cost of the Gelatin as the method has not changed.

Continuing to use the 3000-lb-plus string of tools in our blasthole drilling, an increase in footage per drill-shift of about 18% in hard ground over the lighter tools has been experienced. No appreciable increase in maintenance cost resulted. As the heavy stems wear down they are shifted to drills in the softer ground where the drilling is more a job of mixing than of penetration and weight is not so important.

We were a little hesitant about trying out 12-in. blastholes because with the added burden we did not believe that they would give satisfactory fragmentation in hard ground. We were also dubious about the feasibility of drilling 12-in. hole with 29T drill rigs. However the trial was made with some old 10-in. bits upset to 12 in. and the results were sufficiently encouraging so that an order was placed for 12-in. bits to continue the trials in all types of ground. Results of the trial are shown in Table 1.

Table 1—Comparison of 12-in. versus 9-in. bits.

	12-in.		9-in.	
Type of ground	Performance Hard	Cost	Performance Hard	Cost
Feet of hole in test	1456		2860	
Spacing	36 x 21		33 x 18	
Tons broken	78,840		116,922	
Tone broken per ft of hole	54.1		40.9	
Feet drilled per shift	48.6		79.1	
Drill cost per ton broken		.0213		.0171
Tons broken per lb of powder	3.818		3.414	
Powder cost per ton broken		.0309		.0345
Labor cost-loading		.0039		.0043
Boulder per ton broken	.00595		.00523	
Mudcapping cost per ton broken		.0039		.0035
Total Cost Per Ton Broken		.0600		.0594
Tons per shovel-shift	4693		4549	

Tons per shovel-shift show an increase, however the test has not been extensive enough to prove a definite gain and has not been figured into the cost picture.

Hard facing churn drill bits is a common practice in many types of churn drilling and one that we have tried previously without a great deal of success. The Lincoln Electric Co. recently asked for a trial of some of their special rods in this service and the results shown in Table 2 were obtained.

Bit No. 43—Preheated to 400F and built up on reaming edges and stinger with Toolweld A and O. Bit was then forged and tempered in usual manner.

Bit No. 42—Built up with Stainweld A7 and surfaced with Tungweld C on reaming edges and stinger.

Bit No. 34—Built up with Stainweld A7, reaming edges with Tungweld F and stinger with Abrasoweld.

Bit No. 24—Built up with Stainweld A7, reaming edges with Hardweld AC100, stinger faced with Abrasoweld and Faceweld No. 12.

Table 2—Results of test with hard surfaced churn drill bits.

	Cost			
Bit No.	Rod	Labor	Preheating Forging & Tempering	Total
43	$0.54	$1.25	$5.00	$6.79
42	3.90	1.13	..	5.03
34	3.46	1.88	..	5.34
24	1.42	2.25	..	3.67

	Performance			
Bit No.	Ground	Ft. Drilled	Standard Bit	Remarks
43	Hard	28	25	Normal Wear
42	Medium	58	37	Normal Wear
34	Hard	11	11.5	Weld Broke Off
24	Hard	4	11.5	Weld Broke Off

Due to the small number of bits going through the bit shop at present, sharpening costs are higher than usual so that none of these costs, with the exception of Bit No. 43, appear excessive. In hard ground, breakage of the hard surfacing may reject that type but in medium and soft ground the greater abrasive features may show a saving. Tests will be continued until we have some conclusive results. The Tungweld F and C have small pellets of tungsten carbide in the rod matrix.

This paper was presented at the November 1949 Arizona Section meeting, and is a sequel to the paper by M. M. Stover published in Mining Engineering, May 1949.

Mr. Van de Water is mine superintendent at Castle Dome and an AIME member.

A New Way of Educating Engineers

by Arthur F. Taggart

ENGINEERING is the art of control of men, matter, and energy in timely concert to provide economically the material needs of man.

Control requires knowledge of the thing to be controlled and of proven methods of controlling.

Knowledge of man comprises understanding of what he is and of his reactions as an individual and in the mass to his surroundings and to other men.

Knowledge of matter comprises understanding of its fundamental nature, its methods of combining into multitudes of animate and inanimate forms, the properties of these forms and their various utilities, their sources and availability.

Knowledge of energy constitutes knowing what is meant by the term, its various forms and sources, the laws of convertibility, and the methods of transformation and transmission.

Science is codified knowledge of facts and of the verified general laws that apply to them. Chemistry and physics are the sciences that systematize our knowledge of matter and energy. Mathematics is the principal tool in their application to technology.

Engineering differs from science in that action is its essence and that such action requires knowledge beyond that which science furnishes. This knowledge is encompassed within a large body of empiricism, more or less firmly established, on which methods of engineering control are founded.

The distinguishing characteristic of modern engineering schools, in so far as undergraduate curricula are concerned, is improper fragmentation of this empirical knowledge. We package it in departmental wrappings instead of barreling it as Properties of Materials, Energy Transformation and Transmission, Machine Elements, and the like. These subjects cut across all departmental boundaries. No engineer is educated without knowledge of the basic principles of all.

Engineering practice involves application of science to the limit of its coverage, thereafter the use of established empiricism as far as it extends, then always the exercise of judgment to throw a frail bridge across the gap of ignorance, and finally action. The time factor involved in each of these preliminary steps controls the essential timeliness of the action. The magnitude of the time factor depends upon the amount of study required for each step. The basic aim of undergraduate instruction in engineering should be to lay such a broad and firm foundation of knowledge that study may be adequate within the time limits that economy and competition impose.

The adolescent desire for a label, the desirability of orientation before graduate work, what is presently but to a large extent erroneously taken to be the attitude of industry, and the vested interests of instructors combine to prohibit elimination of specialty courses from undergraduate curricula. But they should be held to the minimum possible.

The attitude of industry is to defer imposition of directive responsibility and delegation of authority until maturity and good performance under direction give reasonable promise of ability to assume them. Large industry states plainly in employing young graduates that it intends to apprentice them for periods ranging from months to years to teach them the details of its practice. It asks that we give the basic training to aid the apprentice's learning and make him worth educating. Small concerns follow so far as they can.

Education for these starting jobs that will at the same time equip men to complete their apprenticeship quickly and derive from it a maximum of profit can be given in four years with a curriculum stripped of repetitiousness and operating detail.

The aggressive extrovert should be urged into industry at the end of four years and be accepted for graduate work only on the strongest possible showing of capacity for research and a persuasive showing of true liking. Below average students should not be accepted for graduate work.

A curriculum consonant with these generalizations follows.

About the article

Curtis L. Wilson says: *This curriculum would be turned down by the ECPD Committee on Engineering Schools. It is not engineering.*

Arthur F. Taggart replies: *It might be well if the present standards by which curricula are judged by the Committee on Engineering Schools be reviewed. I hope that engineers who believe that engineering education should be heavily weighted toward practice in a specific line will so inform and thus support the Committee; whereas those who believe that a broad, basic engineering training which affords its graduate maximum flexibility in practice will agitate for a change in the Committee viewpoint.*

Curriculum

(1) **Human relations.** What is sought here is a foundation for both satisfactory living and successful engineering. Consequently it should receive a generous time allotment. It should certainly be continued throughout the undergraduate course in order to make its teachings as much as possible a part of everyday habit. Specific subject matters are:

Elementary biology, to teach the nature of life.

The humanities in their various aspects, to teach how sensitive and generally sympathetic men and women react to life and to the conditions of their times.

Political and economic history as a record of the responses of man in the mass to his surroundings, natural and man-contrived, and to his leaders, good and bad. The result of these responses is the ever-changing pattern of civilization. Understanding of this fact is essential to the engineer.

English, spoken and written, as essential tools of rapid and accurate communication.

This article is an excerpt from the talk given by Professor Taggart, Columbia University School of Mines, before the MIED at the Annual Meeting, Feb. 12, 1950.

(2) **Science**, as the basis for all technological understanding and prediction, and as the part of the educational fare most readily taught in college, is given major weight. I suggest a four-year stretch of physics, chemistry, and mathematics, integrated and intermingled as completely as a competent teaching staff can do the job. This means neither as a chemist would teach a prospective chemist nor as a bored master physicist treats students with different aims than his. The perfect teacher would be the man in whom knowledge of the three sciences had blended into a universal tool, which he aspired to make ever sharper by using it to fashion similar tools for his students. Such a man would teach slowly and thoroughly; would confine himself to the relatively few broad, basic generalizations, and concentrate on demonstrating the universality of their applications; he would not go much beyond what our curricula today purport to cover. He could, and even might, save us from publishing in our catalogs the ridiculous paradox presented by a listing of Physical Thermodynamics, Chemical Thermodynamics, Engineering Thermodynamics, Chemical Engineering Thermodynamics, and Metallurgical Thermodynamics. In any event he would require and should be allotted at least a period a day for the four years.

Elementary geology and geography should be included here, both taught from the utilitarian standpoint of materials supply rather than as preparation for professional practice.

(3) **Basic Engineering.** There are perhaps half a dozen subject fields, blending basic science and empiricism, which as a group constitute the framework of engineering. They are presently packaged as Strength of Materials, Fluid Flow, Electrical Machinery, Heat Transfer, etc. Accelerating understanding is rapidly moving them from a largely empirical to at least a semiscientific status, and justifies a broader grouping. At the same time, it is becoming increasingly necessary for the engineering student who aims to be more than a narrowly-skilled technician to acquaint himself with all of them. Instruction can be made more efficient and the time necessary for such broader acquaintance can be saved by some such regrouping of subject matters as: Properties of Solids, Properties of Fluids, Static Structures, Machine Elements, Industrial Energy Transformations, Instrumentation.

Properties of Solids would start with a summary of what the physicist and chemist know of atomic arrangements in solids in relation to their effects on the properties of the solids and to

their control. The metallographist and, in his wake, the structural petrologist would follow with elucidation of the relationship of crystal structure to the physical and chemical properties important in engineering. Whereupon Strength of Materials would begin to move from the cookbook toward the scientific realm.

Properties of Fluids are now taught in a heterogeny of courses and departments. The most that can be said for the arrangement is that it offers unnecessary employment to a lot of professors. From the students' point of view it means that to get an over-all picture of the separatory, energetic and force aspects involved, he must waste many semester hours and listen to endless repetition. The gist of the subject could be taught in a fraction of the time. Much of the specific empiricism now taught is outmoded before the student is ready to use it.

Static Structures and Machine Elements deal respectively with the engineering arts of resisting and applying forces. As such, they are involved in one way or another in all engineering practice. Admitted that many of their problems are highly complicated and are jobs for specialists—which merely means that they have not yet made much progress from pure empiricism toward scientific understanding—they nevertheless contain sound cores of basic knowledge readily comprehended by the engineering student. These cores should be prescribed for and presented to the student in a reasonable time.

Energy Transformation differently packaged for civil, electrical, mechanical, etc., engineering is studied today by all engineering students. No one receives a comprehensive picture and none, consequently, are given the basic grasp of the subject that all should have. Again, specificity defeats the true aim of education.

Instrumentation is not only the evolving tool for most efficient operation of machines and processes, but, short of revolution in national thought and political policy, is a principal defense of capital. On either ground, full knowledge of its methods and possibilities should be taught to every engineering student.

Each of these subjects involves more of empiricism than of science in the present state of knowledge. Each is making giant strides which compress acres of charts and yards of empirical equations into a few expressions dimensionally sound and of wide application. At the same time, further technological advance produces other crops of nomograms and equations with rationalizing constants. But this is the essence of engineering method. It should be so taught by competent engineers, confident in their capacity to select the certain from the speculative, willing to concentrate on the nature of the basic and to forego the lure of the specific, concerned with implanting a similar viewpoint in their students.

(4) **Specialty subjects.** I would limit these to ten semester points in the senior year. With the background of scientific and basic engineering subjects proposed, each department could offer in that time reasonably comprehensive and rigorous courses which illustrate the general practice of its field, and which would justify its certification that the graduate was prepared to enter the field as a beginner with the promise of quick learning and eventual competency.

I believe further that the student who wished to elect equivalent point values in science or to spread specialties between different departments should be permitted to do so, and upon satisfactory completion be granted a Bachelor of Science degree undesignated as recognition that he prepared to enter upon his novitiate.

(5) **Graduate work.** The only justification for advanced work, if I have correctly interpreted employer attitude, is to train for research. The requirements for this activity are all of those human qualities that the employer sets up, raised to a power, with the one qualification that probably, on average, initiative can be shaded in favor of pig-headed persistence, and a little Yankee ingenuity should be thrown in. There should also be love of discovery for its own sake, and a quick and highly retentive memory.

Instruction should comprise rapid, packed lecture surveys of existing knowledge in the scientific and technical fields germane to the subject in which major research is contemplated, instructor analysis of research methods and tools in the same field, and preliminary student practice with the more important of these. Additionally, the student should perform a largely independent research—simple for a Master candidate and distinctly difficult for the Doctoral candidate—to which the instructor contributes no more drive, ideas, or manipulatory aid than is consonant with the concept of independence.

The idea of the program herein proposed started from analysis of personal experience. It was not too long after I started professional practice before I began to wonder why the engineering books that I had studied and the careful lecture notes that I had taken in college never seemed to help on the problems that confronted me. I had been stuffed to the craw with engineering courses, mostly by able engineers, if reputation is to be believed. These men certified that I had absorbed what I was supposed to and graduated me. Two of them even hired me afterward at various times. But not even they ever gave me jobs that those notes covered. Consequently, I threw the notes away.

As a matter of fact, the only specific engineering tool that I learned in college and retained to use is the slide rule.

Now, after thirty-odd years of professional practice, I can truthfully say that no employer has ever been willing to pay me even a living wage to do any of the things that were in those early books or notes. Always what they would pay for has been to start solutions of problems that were not mentioned in the engineering books at all. In this the dog-eared books that have been my salvation are successive volumes of sound, reasonably elementary treatises on chemistry, physics, and mathematics.

Admitted that mine has all been consulting work. It has also been work which, except for certain specialized phases, should have been done by company engineers. It could have been done by them and would have been done better and with much less effort by me with the train-that I propose. It is the course I would follow myself if I could start again with present knowledge of readily salable skills.

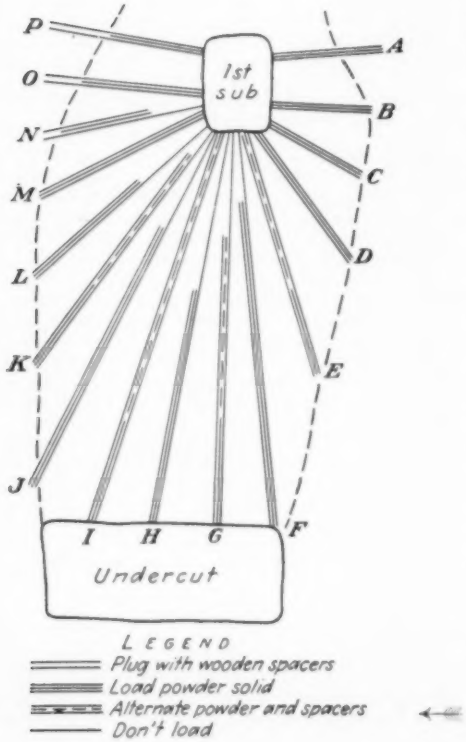
Radial Blastholes

for Drilling an Irregular Ore Body

by W. W. Little

AT the United Verde mine the 9-2D stope on the 4050 ft level is located within the north sulphide mass lying some 800 ft north of the main sulphide ore body. In 1948 it was estimated that this block contained 85,000 tons of ore at 4% copper and 9.2% zinc.

The north ore body is a lenticular replacement deposit in the black schist, extending from about 100 ft above the 3750 level to a distance below the 4500 level yet to be determined. Although the mineralized zone approaches 1000 ft in length, only 300 ft is of economic importance. In width it varies from a maximum of 60 ft to about 3 ft at the north end, and is characterized by pinching and swelling in the ore zone. One of these larger swells is being mined as the 9-2D stope.



Mining costs at greater depth, together with the expected grade of ore after dilution, necessitated a low cost mining method to permit profitable stoping operations. Based on the results obtained in a number of Canadian mines, it was decided to develop the ore body for stoping by the Aldermac or bench method of diamond drill blastholes; however, this plan was subsequently changed and the radial or ring method adopted. Such a method seemed to offer the advantage of large tonnages at low cost, and obviate the necessity of waste fill. This latter point was of particular importance, since the present waste system does not extend below the 3000 level.

Character of the Ore—When development of this north ore body was first planned, the economic value of the zinc was an unknown factor, since previous mill tests on zinc sulphides from other parts of the mine showed that fine grinding was necessary to liberate the sphalerite, which is closely interlocked with pyrite and mineralogically known as marmatite. Although laboratory tests were started in January 1948, zinc values were disregarded when first estimating mineable ores below the 3750 ft level.

Mr. Little, a mining engineer, is mine superintendent at the United Verde mine of the Phelps Dodge Corp., Jerome, Ariz. This paper was recently presented before the Arizona Section, AIME, of which Mr. Little is a member.

A generalized cross section from the hanging wall diorite shows successively bedded sediments, purple porphyry, quartz, sulphide, schist, and quartz porphyry. The chalcopyrite occurs in the sulphides and schist, while the zinc is limited to the sulphides alone, occurring mainly as interlocked sphalerite and some marmatite. Since the ore area is irregular in outline and the schist footwall tends to slack and spall when exposed, it was recognized from the beginning that dilution would be a major problem.

Diamond drilling to delineate the ore areas indicated that the ground would be difficult to drill and break, and this was fully substantiated after development of the block was started in October 1947. Schist drills and breaks quite readily, but the sulphide is hard, fine grained, massive, quite siliceous, and abrasive. New steel bits drill but 2 to 8 in. before dulling, with resultant high costs for drilling and breaking ground.

Fig. 2—Drill hole pattern is varied with the contour of the ore. Diagram shows loading.

The 9-2D Stoping Block—In vertical section, this block extends from the 4050 level to the 3750 level. The stope is approximately 180 ft in length and varies in width from 10 ft to 60 ft.

The sequence of development was as follows:
(1.) Scram drift and connecting finger raises to the undercut horizon are driven. Simultaneously, an access raise started at each end of the block.
(2.) The undercut is silled out from the two access raises.
(3.) Beanhole raise from the undercut to the first drilling sublevel is driven.
(4.) First sublevel off the access raises is opened.
(5.) Beanhole raise to the second drilling sublevel is driven.
(6.) Radial blastholes are diamond drilled from the first drilling sublevel to the undercut, using the beanhole raise as an initial opening which later forms a part of the blasting slot.
(7.) Open the second drilling sublevel while diamond drilling blastholes on the first drilling sublevel.
(8.) Subsequent drilling sublevels are opened and beanhole raises driven to the next sublevel, followed by the diamond drill crews.

The scram drift is located 7 ft above the 4050 level, and connects to the access raises. This limited height does not permit chute capacity, but it is desirable to minimize the tonnage tied up in the sill pillar. Eight finger raises or draw points extend off the scram drift to the undercut, and a 35-hp electric slusher is placed at the south end of the scram, scraping broken muck directly into 50-cu-ft cars spotted under the chute pocket.

The ground being hard sulphide, it was thought that the scram drift and finger raises would stand unsupported, but with secondary blasting the ground began to spall, and after four months it became necessary to reinforce the back. This was done by pouring concrete pillars and lacing from pillar to pillar with 60-lb rails. While this was a makeshift job and maintenance has been rather costly, the ground was held without shutting down the stope for the relatively long period of time which would be required to line the scram with concrete.

Drill Crews—Upon completion of a sublevel drift between service raises, it is cleared of all loose muck and the bottom cleaned with a blowpipe to expose any missed holes before the diamond drill crews move in. The engineering department furnishes the diamond drill foreman with a diagram showing the location, bearing, depth, and inclination of each hole, and the drill crews work entirely from these diagrams.

Normally two diamond drills are used on a sublevel. They are light machines designed for short holes, mounted on a single bar and arm. Each is manned by a driller and helper, experience having determined that the over-all cost per foot of hole is less by utilizing a helper with each machine rather than a driller alone or one helper working between two drills. As a general rule one machine is started drilling the slot at the beanhole raise, while the second is at one end of the stope, so neither interferes with the other.

The diamond bit used is an EX coring bit with a cylindrical preset steel plug inserted in it and held in position by a pin passing through the plug and both walls of the bit. This limits the purchase and inventory to one class of bits, since the coring bit, by the use of the plug insert, serves for blasthole drilling. Although a small percentage of blastholes are cored to determine ore outlines, no segregation of costs are made between drilling with plug bits and with core bits.

Radial Drilling — Although the original plans called for bench drilling, the radial or ring method has been used almost exclusively. It was believed that, due to the weak footwall, the horizontal pillars between drilling sublevels would be weakened unnecessarily by silling out from footwall to hanging wall with the possibility of dropping an entire lift. Secondly, by driving a subdrift through the block rather than silling out, it would be more quickly available for drilling long blastholes. The third factor was the lesser time for moving the drill, as a maximum of four vertical holes may be drilled from any one setup.

The obvious disadvantage of this method is the footage of holes drilled but not blasted; however, the advantages of radial drilling more than

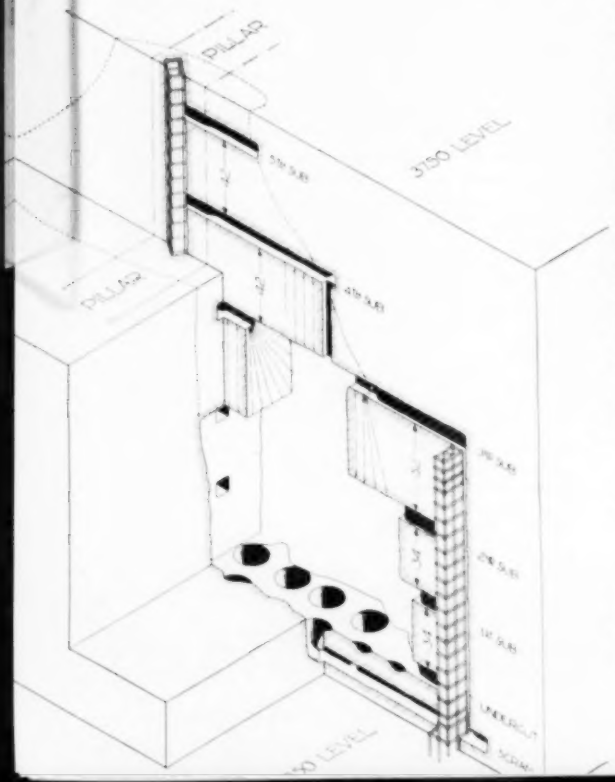


Fig. 1—Radial blastholes, as shown in the isometric, have successfully replaced the bench or Aldermac method.

offset the lesser footage required by the Aldermac or bench system with vertical holes. Present practice is to drive a 6 by 9-ft drift along the long axis of the block connecting the two access raises. The drift is then slabbed to approximately 10 ft in width, which is sufficient to permit the use of 5-ft diamond drill rods.

Holes were first spaced with a burden of 4 ft and a maximum distance of 5 ft between the ends of the holes. These distances have been increased to 5 ft burden and 6.5 ft at the end of the holes, but increased secondary blasting indicates that this burden is excessive and will be reduced in the future.

The first lift taken was 34 ft, the determining factors being the estimated time of handling drill rods, which directly affected the diamond drilling cost, and the deflection of the holes. The vertical distance between subs was later increased to 50 ft, and it is now planned to try 70-ft vertical blocks.

The irregularity of the ore body requires close coordination between the geological department and the drill crews. Although the crews are given a drilling plan for each ring based on previously estimated ore outlines, the completed ring may vary somewhat from the original plan. Holes are added or deleted, depending upon whether the ore zone swells or pinches at that particular cross section, but in general the rings are so drilled that when blasting is completed both walls present a smooth, curved surface. In attaining this wall outline some ore must be sacrificed, and some waste included. While such a practice adds to the dilution of the ore, the final results are better than would be obtained by leaving large bellies of waste hanging on the walls which would later slough, giving even more dilution and further weakening the stope walls.

The Blasting Slot—The initial holes blasted are those adjacent to the beanhole raise. This raise is about 5 ft in diameter and is elongated by blastholes to the full width of the ore, thus forming a slot to which subsequent rings may be slabbed. To insure that the slot will break to uniform dimensions, the burden of ground on the initial holes does not exceed 3 ft, and these holes are loaded solid with no spacers. No difficulty has been experienced with these blasts.

All rings are blasted electrically. Primacord was used to blast the first six rings, but was discontinued when later experience indicated equally good results with electric delays alone. Fragmentation was not as complete as desired, and in an attempt to lessen secondary blasting, the fractional-second-delay blasting caps were tried and their use is now standard practice. It is impossible to conclude that improved fragmentation is due to this type detonator, since the increased height of succeeding lifts, with a corresponding greater distance for the muck to fall, is undoubtedly a major factor. To date eight delays have been the maximum number required.

Although 50% special gelatin dynamite was used for all ring blasting on the first three lifts, later experience has shown that 40% Amogel is entirely satisfactory, and is now used for blasting all radial drill holes. Primary blasting required 12,040 lb of powder to break 52,660 tons of ore, while secondary blasting utilized another 2350 lb.

Explosives Data

Average depth of holes	28.81 ft
Average powder per foot of hole drilled	0.33 lb
Average tonnage per foot of holes drilled	1.45
Average tonnage per lb of powder	
Primary blasting	4.37
Secondary blasting	22.42
Total	3.66

	Maximum	Minimum
Estimated tonnage per blast	3000	500
Pounds explosives per blast	1053	64
Number of rows per blast	4	2
Number of holes per row	19	6

Loading—Several miners on each shift have been trained to load the rings in the prescribed manner (see Fig 2), and these loading crews wear safety belts when working near the edge of the bench.

Costs—The 9-2D stope came into production in June 1948. At the end of July 1949, it had produced a total of 60,130 tons, of which 7470 tons were from sublevel development. The average monthly production was 4295 tons, resulting in a stoping efficiency of 16.21 tons per man-shift, including development and diamond drilling. Direct cost of labor, explosives, and timber amounted to $0.96 per ton, exclusive of diamond drill supplies and bits. During this period 36,217 ft of diamond drill blastholes were drilled at an average cost of $1.08 per foot of hole for labor, carbon, and supplies, or $0.74 per ton ore broken. Although it is not possible to arrive at an accurate cost until the stope has been completed and all muck pulled, the over-all direct stoping cost will probably not be in excess of $1.25 per ton.

Owing to the siliceous character of the ground in the hanging wall sulphides, diamond drill efficiency has been rather low. For the period referred to above, footage per drill-shift averaged 45.5, and the average footage drilled per bit was 67.7.

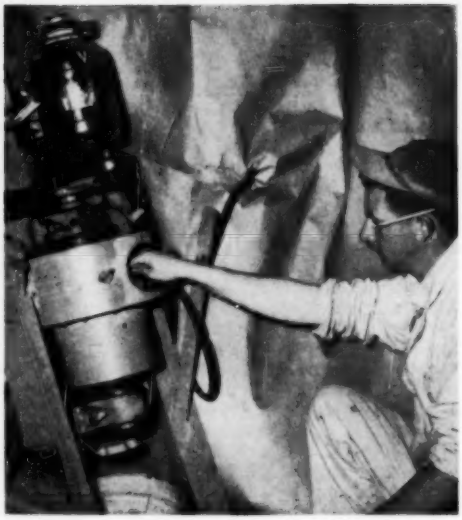
Adverse Factors—Some of the factors which resulted in less efficiency than originally estimated will be corrected in the future for areas mined by this method. By locating the scram drift only 7 ft above the level, it was necessary to load directly into cars. Chute room for four or five cars is highly desirable. Since considerable secondary blasting is essential in the scram drift, a lower over-all cost would be effected by initially providing support for the back and walls as well as the finger raises. Some dilution results from sloughing of the rather weak schist footwall. It is believed that in future operations it will be better not to clean the ore to the footwall, but to leave about 4 ft, in an attempt to hold the waste in place.

Conclusions

Although regularity of an ore body and strong walls are considered requisites for this type mining method, neither condition is fulfilled at the United Verde mine. While this has resulted in more than the normal amount of dilution, the results show that by careful planning and attention to detail, such areas can be mined economically. A relatively large, even flow of muck is assured at a reasonably low cost.

Low Cost Centrifuge Versatile in Laboratory Use

by Earl L. H. Sackett



THIS laboratory centrifuge is unique in its low cost and ease of construction, its particular adaptability to problems encountered in an ore dressing laboratory and its simplicity of operation. It will treat a sample of the size ordinarily used in a laboratory flotation test. The centrifuge was designed at the Baroid Sales Division of the National Lead Co., Malvern, Ark., and built in a local machine shop to serve as a laboratory machine which would produce results comparable with those obtained in the large scale centrifugal deslimers in use in our Magnet Cove mill. The apparatus has served quite well in this capacity and has also proved effective as a deslimer on a variety of ore dressing problems. The cost of making this machine was relatively low and it is simple to operate. It is strictly a batch machine, but when operated under proper conditions, gives results which agree quite closely with large continuous-flow machines of the same type. Effective separation of minus 5 and 10 micron material is made in an undispersed pulp in a short time.

A ground ore pulp, in the density range of usual flotation feed, is fed at a constant rate to the revolving centrifuge cylinder and the unsettled "slime" fraction flows through and is discharged through the effluent ports. The settled portion is retained in the cylinder and is washed out, after stopping it, by tilting the machine and applying a small high pressure stream of water through the effluent port holes. The completeness of separation of a given size fraction is governed by the number of times the material is passed through the machine—usually two to four. The size of the fraction overflowed is governed principally by the speed at which the cylinder is operated and is, of course, inversely proportional. Rate of feed, which governs retention time, also has some effect on size of material overflowed. The machine as shown here has three practical cylinder speeds of 1000, 1700, and 2600 rpm. A greater variety of speeds than is possible with the drive shown here could be provided by electrical control or by use of a Reeves-type drive.

In making a separation the cylinder is first filled with water through the feed line while revolving at the chosen speed. Then the feed line valve is closed and the pulp to be treated is placed in the feed tank. Air escaping back through the feed line keeps a finely ground pulp sufficiently agitated until the operator begins to feed it within a period of about twenty seconds. The feed valve is then opened wide; the feed enters the machine and the separation is made; and the effluent, containing rejected slimes, is collected in a bucket. After turning off the motor and changing buckets under the effluent discharge before the cylinder comes to a stop, the retained material is washed out as shown in the photo.

Mr. Sackett is assistant chief metallurgist, Board Sales Division, National Lead Co., Malvern, Ark., and an AIME member.

The data below illustrate a typical separation made on an undispersed pulp of ore ground to about 90% minus 325 mesh. The machine was operated at 1700 rpm and the separation made in four successive passes. Time required for the entire operation was twenty minutes. The rejected effluent constituted 23% by weight of the feed.

Micron Size	Percent Rejected Effluent	Percent Retained Product	Percent Weight Rejected
Plus 15	8.6	50.2	4.9
Minus 15 plus 10	3.8	16.1	7.2
Minus 10 plus 7.5	5.1	12.3	11.2
Minus 7.5 plus 5.0	15.0	10.6	30.0
Minus 5.0 plus 2.5	24.4	6.1	54.8
Minus 2.5	43.1	4.8	73.1

Review of Progress in the Caving of Asbestos Ore

Asbestos ore is hard, well intersected by free fissuring but not completely enough to avoid heavy secondary blasting, and is a more valuable ore than usual for caving. These factors have resulted in deviations from standard caving practice in details but not in principles.

by Gerald Sherman

MINING asbestos ore by caving at Thetford Mines, Quebec, has been described in the Transactions of the Canadian Institute of Mining and Metallurgy in papers presented by the staffs of The Asbestos Corp. Ltd., in 1934, and Johnson's Co. Ltd., in 1941. The production by caving prior to 1949, in round figures, has been 14,500,000 tons of ore by The Asbestos Corp. and 5,500,000 by Johnson's Co. They are the only caving operations in asbestos at the present time. The Jeffrey mine of the Canadian Johns-Manville Co. at Asbestos, 40 miles southwest of Thetford, is well along in preparations for caving, and The Bell Asbestos Mines Ltd. at Thetford Mines, will follow their examples later.

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AIME San Francisco Meeting, Feb. 1949, and New York Meeting, Feb. 1950.

TP 2828 A. Discussion (2 copies) may be sent to Transactions *AIME before May 31, 1950. Manuscript received Nov. 2, 1949.*

Nearly all of the asbestos produced by the western hemisphere is mined in those districts of Canada. Since it has been the subject of a number of technical papers, this contribution is offered as a review of progress. Some introductory information seems appropriate, for few members of the American Institute of Mining and Metallurgical Engineers are familiar with the mineral, or how asbestos deposits differ from others in caving characteristics.

Asbestos is found in peridotite, a basic intrusive rock irregularly altered to serpentine and erratically fractured, well enough to cave in bulk, but leaving many blocks of solid ore of all sizes scattered through the deposit.

Chrysotile is the variety of asbestos produced in Canada. Its fibers are the most valuable commercially, having a good proportion of long fibers, strong, silky, flexible, and suitable for spinning. The fibers occur in all lengths, from the microscopic to 3 in., or more, the shorter are more abundant. Their values depend on length combined with other qualities noted above. They occur as veinlets filling fissures in the peridotite, with crystals compactly arranged in parallel across the cracks. They are sometimes lightly frozen to the walls but the partings are distinct, often marked by particles of magnetite adhering to the fiber ends. Sometimes they break away from the rock in slabs an inch or more in thickness.

In wide fissures the fibers are not always continuous, the "broken fibers" being separated by lines of parting with occasional scattered grains of magnetite, which is an injurious mineral to be eliminated when possible. In addition to the chrysotile "cross fiber," some "slip fiber" is found in fault fractures in limited quantities of less valuable character.

With the exception of the Asbestos Corp. ore, in which the content is said to be more uniform, asbestos is eccentrically distributed through the peridotite. It is the general impression that the higher grades are more often found in hard rock. There are rich pockets and leaner areas which range down through various grades to minor barren sections. Wholesale mining is necessary to make sure of recovering all the high grade for there are no reliable indications of its existence. Ore selection only appears in constructing assay-grade limits for caving.

The mill yield of the Thetford ore body in 1934 was 6 to 10 pct of the ore milled, made up of all classes, with occasional runs up to 15 pct or so.

Quotations for Canadian asbestos* are as follows:

	Per Ton
No. 1 Crude (¾ in. and longer)	$896 @ $960
No. 2 Crude (⅜ to ¾ in.)	350 @ 545
Spinning (less than ⅜ in.)	204 @ 378
Other classes, price diminishing with length.	
Shorts or "refuse"	24 @ 46

* From *Eng. and Min. Jnl.*, Nov. 1948. Asbestos of even superior quality for electric insulation is found in Arizona limestone. It is white, containing very little combined iron, but occurs only in small quantities.

The pockets of high-grade ore in open pits containing crude asbestos in unusually close assemblies of veins, are blasted with care to avoid breaking down the long fibers. The rich portions are sorted out and cobbed by hand to free them from rock and magnetite and are separated into No. 1 and No. 2 Crude grades for market preparation.

The pit ore and the rejects from the above processes are dried and milled by alternate stage crushing and screening to release the fibers (deliberately fluffed up in milling), which are lifted out of the sands by pneumatic suction. Wood splinters which are rigidly excluded in specifications would also be picked up with them. The use of timber in mining is therefore not permissible.

In the final treatment, fibers are segregated by Quebec Standard screens into many specific grades.

This page of Transactions AIME is a continuation from p. 402. The missing pages appeared in Mining Engineering.

Fig. 1—Pattern of fracturing opened by subsidence of supporting ore. Natural fragmentation.

Fig. 2—Large block of untouched ore isolated by blasting about it.

Fig. 3—Large fragments of ore jammed in a draw hole above a grizzly.

Combinations of certain grades for special purposes are frequent. There must be close cooperation between the milling and sales departments, and the purchasers.

There is an increasing demand for asbestos, and now shorter fibers are being used than could have been sold a few years ago. Milling equipment has been adjusted to recover them, which will increase the profit and total quantity of fiber sold, but slightly decrease the average length and value per ton.

The similarity of the analyses of asbestos and peridotite prevents the determination of asbestos content by assay. It is estimated by "reading" diamond-drill cores from holes drilled for test. Veinlets of asbestos are measured, and other evidence appraised visually by experienced observers. As the asbestos distribution is extremely erratic, with rich and barren spots, the test holes must be closely and uniformly spaced to obtain enough samples to make up a true average. In doubtful areas holes have been drilled at the corners of 80 ft squares, but that might not be the minimum limit, for marginal ores. In proven ore bodies, accurate estimates of grade are not too important, but in locating the boundaries of an ore body, or in new low-grade deposits, the drill cores are occasionally milled in laboratory tests for a check on the readings.

Thetford Mines Deposit and Ore Characteristics

The Thetford ore body is pear shaped and about 2200 ft in length and 1300 ft in maximum width. Ore is now being mined more than 1000 ft below the original surface, but its ultimate depth is unknown. It was mined in open-cast pits to a depth of 350 to 400 ft, the usual economic limit of surface mining there.

The ore is too greatly fractured to be mined safely in open stopes, and timber underground is prohibited. By a process of elimination, caving was selected for the method of mining in spite of ore losses, dilution, and deterioration of fiber by attrition in its passage to a draw hole. Caving from a high column of ore reduces the proportion of ground support necessary for miners' protection to a point where the cost of steel and concrete per ton of ore mined is not excessive.

The fragmentation of ore in subsidence has a controlling influence on the efficiency of caving and the cost of mining. There are many open fissures in asbestos ore as well as potential planes of weakness.

Patterns of fragmentation by free fissuring are shown in figs. 1 and 2 of surface exposures undermined by subsidence from below. The blocks so isolated may be broken in the process of drawing, but many are hard and arrive at the draw holes unchanged, possibly with their corners rounded. One mass 60 ft in length appeared, covering several draw holes.

The occurrence of serpentine, widely but irregularly distributed, is characteristic of the district. It is found on the faces of fracture planes and in pockets of loose fragments, wedge-shaped and slippery (fish scales). It is favorable for caving but makes a treacherous formation for development, particularly in raising.

There is a tendency for the fines to run out from among the larger boulders which frequently become locked together high above the grizzly level (fig. 3).

Drifts in ore 8 ft or less in width can usually be driven some distance in advance of steel sets which may be required later. If the roof is caught up before movement starts, the pressure is seldom great. The ore does not swell.

Close lagging with steel is more difficult than with timber which can be easily cut to fit. Concrete pumped or blown in is most effective in completely filling the space outside the sets.

The peridotite which surrounds the ore body is generally sound, and the shaft stations on the first level stand without support. Developments on the second level indicate the probability of weaker ground, and more steel may be needed. A trial of more massive concrete without steel is being made.

At the Canadian Johns-Manville Jeffrey mine the country rock between shaft and ore was badly broken and extremely heavy. Sets were spaced at half the customary distances, and throughout long sections, top forepoling and sideboards were necessary to hold the ground. Heavy steel beams used for spiling were bent or broken. Two parallel haulage drifts were driven to avoid traffic congestion, and will permit repairs without stopping production. The ore is well fractured but is not so heavy.

The area opened is but a small fraction of the total and may not be typical.

Caving Asbestos Ore at Thetford

In 1931 serious consideration was given to the possibility of profitably caving the asbestos ore at Thetford. The Asbestos Corp. was able to successfully transfer a highly specialized mining method developed for physically weak ores to a new and untested formation.

The general theory of caving requires no description here. Asbestos practice follows "block caving" as at the Miami Copper Co. in 1932, except that ore is drawn from the sides of the grizzly drifts. No ore pillars are left between blocks.

The essential differences from ores usually caved that affect the underground mining of asbestos are:

1. Asbestos ore is harder, and although natural fragmentation in the process of drawing is adequate

for caving, it is not complete enough to eliminate large masses of solid ore. The proportion of oversize boulders which reach the draw points is high, resembling the Climax molybdenum ore rather than the "porphyry coppers."

a. More work is necessary at the draw holes to reduce the ore to handling size, at considerable expense. The daily production per block is low.

b. Delays which occur in the blocking of draw holes cause serious interruptions in systematic drawing.

2. Steel and/or concrete must be used in ground support instead of timber. It is more expensive but stands up better in grizzly drifts.

3. To fulfill market requirements, certain grades of asbestos must be produced. Several blocks must be in production from which to select and blend the asbestos to obtain a suitable current product. For this, flexibility in rates of draw is necessary.

4. It is possible to draw slowly from any block, or at the maximum rate, as desired. The essential strength of ore fragments is such that they are not materially crushed under accumulating pressure.

5. The ore is more valuable than that from a typical caving deposit. More care should be taken, and expense incurred, if necessary, to obtain high recovery. Unfortunately, for lack of accurate preliminary sampling, records of ore recovery and dilution cannot be calculated.

There is one condition in the Thetford ore body peculiar to that district. It is owned by three companies in adjacent tracts, within vertical boundaries, mined independently from three different elevations, two by caving, the third still from an open pit.

Company	Method of Mining	Elevation of Production Level above Sea, Ft.
Bell Asbestos Mines	Open pit	Pit floor, 730
Johnson's Co.	Caving	1st level undercut, 480
Asbestos Corp.	Caving	1st level undercut, 565 2nd level undercut, 165

Through the courtesy of the Asbestos Corp., Johnson's Co. was permitted to observe its caving operations after several years of production and were given information on block dimensions and the designs for steel and concrete construction in grizzly drifts, etc. Having the benefit of the example of a successful caving operation in the same ore, it was decided to follow the same methods.

The following is a description of Johnson's Co. practice, with which the author has been associated. It does not cover parallel investigations and improvements made by the Asbestos Corp. which, it is hoped, will be described at some later date.

Block Development

A standard block is 160 by 160 ft by 300 ft in height. To fit the ore outline it may be considerably longer. Local extensions in height are drawn with the rest.

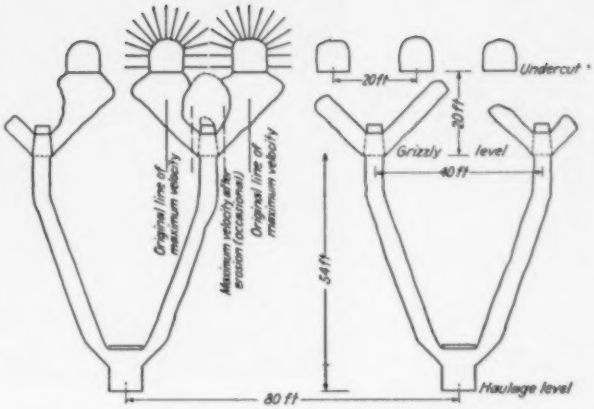

Fig. 4—Section through undercut, grizzly and haulage drifts, showing "ring drilling" for undercut.

The main development level is the grizzly drift (elevation 460 ft above sea level) which is directly connected with the shaft for service, supplies, and ventilation. The haulage level is 54 ft below.

The undercut is 20 ft above the drift, at elevation 480 ft, from which the caving height is measured (fig. 4). Corner raises and fringe drifts are driven from the undercut to the top of the ore, the latter spaced at 40 ft vertical intervals.

The grizzly drifts are driven 40 ft apart, and the draw holes and grizzlies are spaced at 26 ft 8 in. intervals in them. Finger raises are driven upward from both sides of the grizzlies and belled out to reach the undercut 10 ft from the drift centers, making the draw-hole spacing in the undercut 20 ft by 26 ft 8 in.

In the 40-ft section between grizzly drifts at the beginning of the draw, there are two draw holes in the undercut, each 10 ft from the drift center and 20 ft apart in the middle. It is found that by erosion, or blasting, the draw-hole centers are frequently shifted nearer the drifts, that is, to 6½ or 7 ft instead of 10 ft, leaving a high ridge 26 to 27 ft wide between them. The drift pillars occasionally are worn down to the concrete or steel, and arches may connect across them (fig. 5). This seldom occurs across the 27-ft ridge, which cannot be reached easily from either draw hole in attempts to bring down a hanging roof. In the first case, drawing from one hole may undermine an arch over the other, but there is little mutual assistance to be expected across a 27-ft gap, or the 26 ft 8 in., between draw points in the drift. Close spacing is obviously more effective, but there are practical economic limits in that direction.

The haulage drifts are parallel with the grizzlies, and 80 ft apart. The loading chutes are directly opposite across the drift, each receiving ore from a grizzly through a haulage raise.

Fringe drifts and corner raises are driven to facilitate caving promptly, to direct the planes of breaking, and to avoid projections of ore that may overhang from the next block and ride down on the fill (fig. 6). It is particularly important to develop a clean break on a property line.

Quebec safety regulations do not permit untimbered vertical raises more than 40 ft in height.

Fig. 5—Top of draw point showing eroded pillar over grizzly and connection with opposite finger raise.

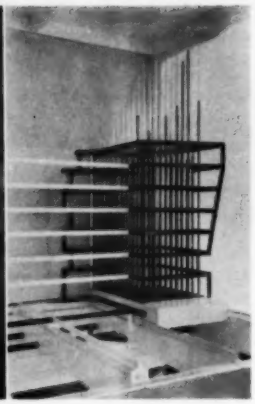

Fig. 6 — Drawing model of block No. 1 sand in glass tubes indicates ore in draw points still to draw. Offset service raise, in left, rear. Inclined corner raise at fringe drifts on footwalls at right. Developments completed painted black.

The corner raises and fringe drifts of the first two blocks were driven from service raises which are in 40-ft sections, offset at the levels of the fringe drifts. A central service raise was afterward driven upward, from which all later blocks were developed by driving fringe drifts from it. Corner raises were driven through from drift to drift to connect them. These could usually be broken through without support, for no more than a hole is needed.

A small engine and cage hoists men and supplies from the grizzly level, and development ore is dropped through a chute compartment to a haulage raise. Many of the fringe drifts had to be driven in advance of requirements, but the gains in efficiency, cost, ventilation, and safety are satisfactory.

The undercuts have generally been broken by ring-drilled blastholes from 8-ft parallel drifts 20 ft apart. The drills were set up on long horizontal bars in the centers of the drifts, and the holes were drilled radially through the 12-ft pillars and over the drifts. Undercuts have recently been broken by horizontal, 40-ft diamond-drill holes charged with powder and detonated from both ends when possible. This breaks a shallower undercut that appears to be adequate in most ore and obtains some economies in drilling and blasting. It should be described when more data are available.

The steel sets for grizzly drifts, as built by Johnson's Co., are shown in fig. 7. It is a copy of the Asbestos Corp. design differing, if at all, in minor operating details. This is still standard for grizzly-drift caving. Steel alone was found to be too weak so it was reinforced with concrete (fig. 8).

Most of the grizzly drifts can be driven without even temporary propping. Since the steel is cut to standard lengths there must be some clearance for erection in the drift section, which, with the usual overbreak, makes room for an effective thickness of concrete.

The sets are bolted together, and a concrete floor laid, anchoring the posts in position. Plates are bolted to the inside flanges of the H-beam posts, etc., to serve as removable forms. Concrete, blown in by a pneumatic placer to fill all space between steel and solid ore, stiffens the steel sets materially, although there is not the concrete-steel bond obtained in reinforced concrete.

Drawing Ore

The cost of drawing ore at Thetford is high. A great deal of secondary blasting is necessary to get the many large boulders through the draw holes and grizzlies. The draw holes are 16 sq ft in area, and the grizzly bars are 16 in. apart.

To break down the arches that often form too high above the draw holes to be easily reached from below, requires superior judgment and experience. Time must be allowed for necessary safety precautions. Production there is suspended until the arch is broken.

Records for a month have been assembled in table I of the approximate figures of consumption of labor and explosives for a comparison of efficiency data.

Table I. Drawing Ore, Segregation of Labor and Explosives

	Labor, Pct	Explosives, Pct
Breaking down arches:		
Drilling	21	32
Bombing	17	14
	38	46
Working ore through:		
Draw holes	38	39
16-in. grizzlies	24	15
	62	54

It has not been possible to change the standard grizzly-drift layout and design for the first level to increase the draw-hole area or shorten the spacing between draw holes.

The advantages of larger openings for the passage of bigger boulders are obvious. The remedy for the treatment of high-hanging arches is not as simple. When, at Miami, a pipe through broken ore brings capping in prematurely, that draw hole is blocked off and those immediately surrounding it are drawn in the hope that one or another will break the pipe and the flow of ore be resumed. If the holes are too far apart they will be of less mutual assistance.

It is thought that similar disturbances of ore above the grizzly level by the drawing of adjacent holes may break an arch, but they must be close to be effective. Increasing the draw-hole area is not likely to have much influence on the formation or breaking of high-hanging arches.

The correct draw-hole spacing is still open to question for particular conditions. The only objection to close spacing is the block-preparation cost. There are a number of advantages which are difficult to estimate.

1. Theoretically, closer spacing will produce more ore from the volume to be drawn with no greater dilution.

2. Less ore will be left above the grizzly drift in ridges around the draw holes. What more can be drawn will increase the tonnage from the block and reduce the cost per ton.

3. There is a chance that the spacing can be close enough to reduce the number of hanging arches, and consequently the cost of bringing them down.

The remnants of ore drawn from the ridges about the draw holes are expected to be recovered from the lower block, if there is one. What is left lies above the steel and concrete of the grizzly drift. The probabilities of recovering that ore are not good and it would have little present worth. However the value of asbestos ore warrants an investigation of means of recovering more of it.

The maintenance of grizzly drifts has cost less than expected. At least two thirds consists of repairs to the grizzly sets (cabanes) caused by blasting or the erosion of drift pillars over them, which sometimes exposes the steel. It is assumed that repairs will be in a proportion to the ore drawn. That due to blasting will be in direct proportion to tonnage. General repairs will be light in the beginning and will increase in time until replacement is extensive. There have been few replacements of cabanes to date, and only one drift collapsed from general pressure. In that case it is thought that undercutting left a pillar standing too small for the weight thrown on it.

Ore Recovery

The first block of Johnson's Co. was estimated to contain 1,112,700 tons of ore. According to the records, 1,074,000 tons were drawn, leaving some

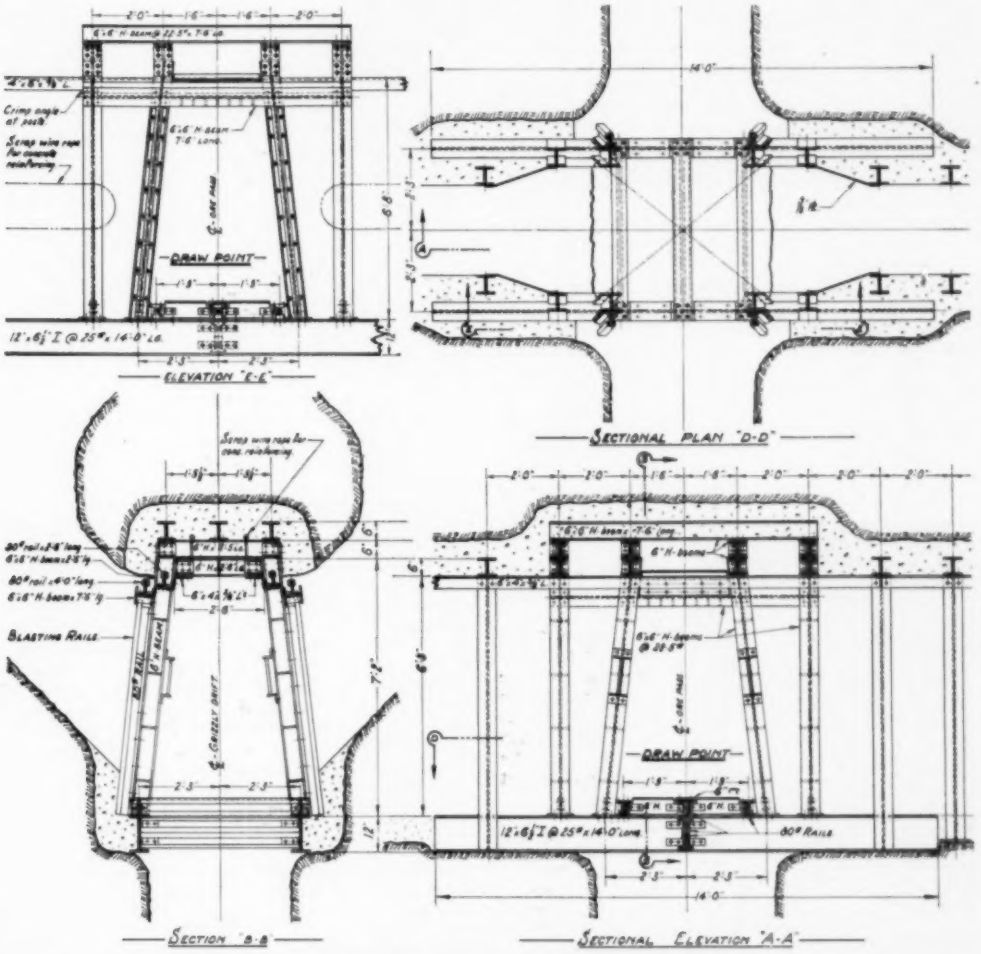

Fig. 7—Design of steel and concrete for grizzly drift, Johnson's Co.

Fig. 8—Steel grizzly drift set at cabane showing grizzly bars and top of haulage raise. Floor concrete not poured.

thousands of tons of known recoverable ore. Without knowing the original asbestos content, it is impossible to estimate either asbestos recovery or its deterioration by attrition in its transit to the draw holes.

The irregularities of ore fragments in size, added to great variations in the rate of drawing because of secondary blasting and interruptions because of hanging arches, obstruct effective attempts at systematic drawing. The conditions are distinctly unfavorable for high-ore recovery.

In ore as valuable as the asbestos ore caved, it is wise to thoroughly investigate the conditions essential for optimum recovery.

Height of Column of Ore Drawn

It is believed that the height that yields the optimum profit at Thetford is about 300 ft. That is the height* at which the curve of cost per ton in relation to height, flattens off. Beyond that height, the economies in cost are trifling. Theoretically, the curve should represent the profit/height ratio but, in the uniform low-grade ore of Miami, the curves would practically duplicate each other.

* F. W. Maclennan: Miami Copper Company Method of Mining Low-grade Orebody. *Trans.* AIME (1930) 91, 39.

At Thetford, where the cost preliminary to production and the ore value are both greater, the profit/height curve would be similar, but there are not sufficient data for its accurate construction.

To offset the gain in mining cost that might be obtained by a higher column caved, there are two adverse factors: the cost of grizzly-drift repairs, and the effect of height on ore recovery and dilution.

Fig. 9—Johnson-Bell property line, black. White line indicates outcrop of break on which Bell ore slipped.

The cost of grizzly-drift repairs has averaged a little less than 4 cents per ton, but repairs during the last third of the ore drawn, cost at least one cent per ton above the average. The trend indicates a further cost increase for a block above 300 ft in height.

The proportion of ore lost, and top dilution, should not theoretically increase with height. Side dilution will be in proportion to the areas exposed to dilution. To preserve the ratio of ore to side areas in high blocks, the bases must be increased, possibly to an unwieldly area for drawing. It is also believed that side dilution will then increase faster in direct proportion to the height, because, during ore (and tailings) movement while subsiding, there is less resistance to pressure from capping in the previously mined block beside it.

It is generally assumed that uniform, systematic drawing is desirable and that it is attempted. It is practically impossible at Thetford. The cross currents that occur in the most uniform ore, systematically drawn, will become more eccentric when fragmentation is as irregular as at Thetford. Local interruptions in drawing caused by hanging arches still further obstruct uniformity, as retimbering does at Miami.

Although average results will ultimately follow principles, a loss of drawing control is more important in valuable ore. High columns are more vulnerable and the consequences more serious.

Although a moderate height has been recommended for caving asbestos ore, other conditions may affect the height selected. The second level of Johnson's Co. will draw from blocks 400 ft high, and Johns-Manville's first level will be 350 ft below the open-pit floor.

Trespass on a Property Line

In the early thirties, the three operations in the Thetford pit were not too far apart in elevation. The Asbestos Corp. went underground first and its greater production carried its active levels farther below the others. Johnson's Co. followed it down at a slower pace, and the Bell Asbestos Mines still more slowly.

The bottom of the Bell pit is approximately 250 ft above Johnson's Co. undercut and 565 ft above the Asbestos Corp. undercut on its second level (fig. 9). It is 315 ft between the drawing levels of the latter companies.

There was no barren capping within the pit areas when caving started, but a backfill of mill tailings has been deposited on the subsiding blocks and maintained at the Bell pit level, with a surcharge above, to assist in supporting the vertical faces of ore in place on the property lines. It may be considered an artificial capping to be treated as customary when drawing ore and avoiding dilution.

This practice was introduced by the Asbestos Corp. and followed by Johnson's Co. It has been generally successful to date, except where a major structural break dipping from the Bell pit, carried some of the Bell ore over the line into the Johnson's Co. workings. It was perhaps complicated by a temporary lack of sufficient resistance from the tailings, as occurs in the movement down to follow subsiding ore, which is frequently in surges.

The loss of ore was appraised after a joint examination, and compensation assessed in accordance with a method prescribed in a previous agreement.

Ventilation

As no rock gases have been discovered at Thetford, and as there is no decaying timber, ventilation

is mainly for the quick removal of powder gases from working places and their replacement by fresh air. If that is accomplished, there should be a good working atmosphere in the mine. Ventilation is a matter of efficient distribution rather than of the quantity of air forced into the mine.

In the cold winters of Quebec, ice will form in a permanent downcast opening. In the small area of Johnson's Co. mine there is a safety exit in depth through another mine, which can only be used in emergency. The shaft is partitioned between the skip and cage compartments (fig. 10). A ventilation fan is situated on the grizzly level near the shaft, which draws air down one half of it to the bottom (haulage) level, up through the workings, to and out through the grizzly drifts (where most of the blasting is done), to the fan, and forced up through the other half of the shaft to the surface. Doors at the two shaft stations can reverse the air through the shaft, thawing any ice near the collar by air at rock temperature, without changing the current through the mine or fan. It has not been necessary to do so frequently, even in a cold winter.

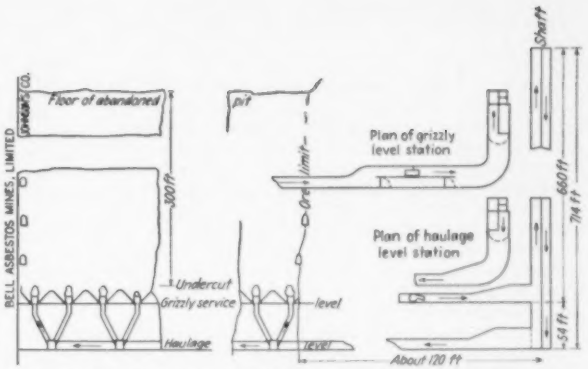

Fig. 10—Diagram of ventilation circuit in Johnson's Co. mine.

Only one air lock at the fan is needed, none in the haulage drifts, but there should be doors or booster fans to direct air to the producing blocks, and prevent short circuiting.

The distances are short, and mine resistance including the shaft bottleneck is not excessive.

In November 1948, gas, thought to have been methane, was discovered in a raise of the Jeffrey mine. It exploded and burned two men rather badly. A small flow from a fissure burned when lit. Methane-detecting equipment is now kept underground. It is unlikely that enough will be found to affect a ventilation system.

At the Jeffrey mine, the undercut level above the scram drifts will be only 350 ft below the floor of the open pit when it is abandoned. The producing area is too large for ventilation from one point (fig. 11).

The advantages of a shallow depth with convenient corner raises can be utilized by laying out independent unit-ventilation areas, each including several caving blocks with a fan of appropriate capacity and two raises, one for a fresh air inlet and the other upcast for the exhaust. They may be corner raises or one could be a service shaft if more convenient.

A pressure system has advantages. It would carry fresh air down to a connection with the haulage level and force it through scram drifts to the exhaust drifts and upcast raise.

All haulage drifts would be in the fresh air circuit fed with air under pressure. Leakage through air ventilation doors supplemented by air directly from the fans would carry air to the main shaft which would be upcast.

The course of exhaust gases would be short and direct.

Transportation Equipment

The haulage equipment (table II) for the three operations is planned for the projected rates of production, and the anticipated fragmentation of the ore as drawn; 4000 tons per day for two 8-hr shifts for the Asbestos Corp.; 2500 tons for Johnson's Co.; and 12,000 (plus) for the Jeffrey mine of the Canadian Johns-Manville Co.

Table II. Haulage Equipment in Caving Operations

Cars				Track	
Co.	Car Cap., Tons	Type	Dump	Gauge, In.	Rails, Lb
Asbestos Corp.	5	Granby	Automatic, to one side	30	56
Johnson's Co.	5	E. Long Special	Automatic, to one side	30	60
Jeffrey mine	10	Rectangular, solid body.	2 car tipple	36	80

Minimum curve radius, 40 ft for the Asbestos Corp.
50 ft for others.

Locomotives				Haulage Drift Section
Co.	Type	Cap., Tons	Volt, Dc	
Asbestos Corp.	Trolley	8-9	220-250	8 ft x 8 in.
Johnson's Co.	Trolley	6	125	8 ft x 8 in.
Jeffrey Mine	Trolley } Battery }	15	250	8 ft 4 in. x 9 ft 6 in. above ties

A small battery locomotive is used by Johnson's Co. for service on the grizzly level.

The Jeffrey cars are designed for strength, and capacity per square foot of drift cross section. Large

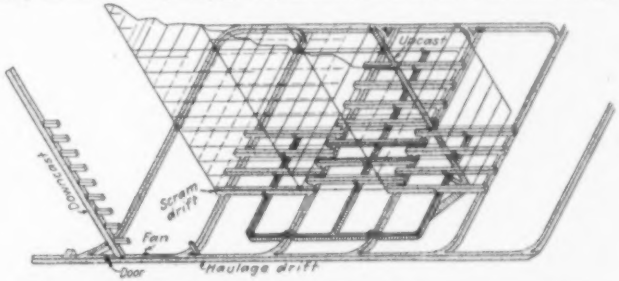

Fig. 11—Diagram of unit ventilation area in Jeffrey mine of Canadian Johns-Manville Co., under pressure system.

boulders will be dropped several feet into them.

The rail weights are for main line tracks. Branch lines for loading ore at Johnson's Co. mine have 40 lb rails.

A heavy copper booster cable was strung by Johnson's Co. from the shaft and cut in at the end of the main line haulage track to permit the use of 125 v current within the ore body. Automatic reels were used in development and found so convenient that their use was continued for haulage through the loading zone. They eliminated trolley wires, rail bonds, and protection for loaders at the chutes.

Loading chutes at Thetford are 48 in. wide, controlled by Cramp chain gates. Johnson's Co. chutes have hinged lips to bridge the gap between chute and car and reduce spillage in loading.

Fine serpentine and fiber on the tracks reduce driving-wheel traction which limits the live train load. Heavier locomotives would increase it.

Hoisting

All mine shafts are of the rectangular type with a large cage compartment, two skip compartments, and a fourth for a ladderway, cage counterweight, air and water columns, and power lines. The dimensions are proportional to production, and the large cages will accommodate locomotives and cars in running condition. The Asbestos Corp. cage has special fittings for lowering long steel sections conveniently.

Underground equipment for mines, crushers, pumps, locomotives, etc., are increasing in unit weights and numbers used. Supply service through the shafts for mines of large production must be planned for the rapid handling of men and supplies, with a margin for increases.

Steel beams are employed for all shaft sets. Johnson's Co. uses copper steel with British Columbia fir, creosoted, for lagging.

At the Jeffrey shaft which passes through badly shattered rock with more than usual overbreak, and which will serve immense ore reserves, concrete is poured about the steel sets in place, flush with the beam flanges of wall and end plates. Johnson's Co. recently sank its shaft 400 ft deeper, concreted in a similar manner, because test holes indicated doubtful ground.

Three sizes of skips are used: 6 tons for Johnson's Co., 7 for the Asbestos Corp., and 10 for the Jeffrey mine.

Two of the three shafts were sunk through considerable depths of water-bearing glacial sands and gravels.

	Depth to Water, Ft	Depth to Rock, Ft
Asbestos Corp.		15
Johnson's Co.	24	94
Jeffrey shaft	35	79

The Johnson's Co. and the Jeffrey shafts were sunk to rock under contract by the Foundation Co. of Canada, which carried down large open wells, inside of which hollow concrete cylinders were poured. The concrete sections are sealed into rock, below which the shafts assume their dimensions. Quicksand in the Jeffrey shaft made it an extremely difficult job.

The concrete cylinders are also used as foundations for the headworks. The Jeffrey shaft continued the concrete cylinder upward in a reinforced section as a self-contained head frame for shaft guides, sheaves, dumping gear, etc.*

* Can. Min. and Met. Bull. (Jan. 1949).

Scram Drift Method of Drawing Ore

The cost of drawing ore through small draw holes and 16 in. grizzlies is very high because of secondary blasting in cramped quarters to reduce large ore fragments to handling size, and because of trouble with hanging arches.

In the scram drift-scraper system developed at Climax, much larger passages for ore are provided for transportation to the underground crusher, which will protect the skips.

The Jeffrey mine, of the Canadian Johns-Manville Co., will follow Climax practice in using draw holes 8 by 4½ ft above scram drifts 7-ft wide. Heavy, powerful scraping equipment and mine cars will deliver the ore to jaw crushers at the shaft with openings 48 by 60 in., which will be the bottleneck for size. Secondary blasting will be greatly reduced in amount, and accomplished under favorable conditions.

Johnson's Co. developed an experimental block of ore for mining through scram drifts. As production will be on a smaller scale, draw holes 7 by 4 ft will deliver the ore to 6-ft scram drifts. Some ore has been drawn which indicates an important gain in drawing efficiency, although, because an underground crusher has not yet been erected, more secondary blasting is done than will be necessary later. Two standard caving blocks on the first level are being prepared for caving by the Climax method. The second level transportation will be designed for better efficiency under new conditions.

It is doubtful if there will be much reduction in the number of hanging arches or the manner of treating them. The arrangement of scram and haulage drifts will permit convenient changes in drawhole spacing. Tests can be made to determine how they influence the number of hung draw holes, drawing costs, and ore recovery.

If there are fewer interruptions in drawing, there might be a gain from more systematic drawing. Trouble is anticipated in allocating correct quantities of ore to each draw hole, which would have the reverse effect on drawing control.

Summary

Since caving began in asbestos ores in 1934 there have been minor changes in caving practice until recently, owing to the inflexibility of the grizzly system using steel and concrete.

As in other mining operations, the use of labor-saving equipment has increased. Jumbos for drill mounting, and power loaders are used in development.

Owing to the high cost of steel and the occasional difficulty of getting it, trials are being made of replacing some of it, at least, with heavier masses of straight concrete. This is mixed and placed by mechanical means.

The major change is in the shift from drawing through grizzlies to larger draw holes and scram drifts that are better suited to handling ore incompletely fragmented in caving.

Acknowledgment

In closing, the author wishes to express his thanks to the three companies for permission to use their operating data, and to their staffs for assistance in collecting it.

Development in the Use of Steel for Underground Support

by F. J. Haller

The need for permanent, fireproof support indicated structural steel sets. Experience over the past six years, involving more than five miles of permanent underground openings, has proved that steel is permanent, safer and, in the long run, cheaper than either treated or untreated wood supports. Excavation is reduced, as are installation, transportation, and handling costs. A number of hazards are reduced or eliminated by the use of steel.

IN 1943, we found, in the new Mather operation, a very unusual and disappointing condition in the footwall rock where all of our main haulageways were to be located. With the exception of a few hundred feet out of the thousands then planned and since driven, all of the openings require support. In general, the ground is not particularly heavy, at least by our standards, but it must be supported and the back carefully covered. Also, our operations require rather wide plat openings (16 to 18 ft) and rather flat curves at the turn-outs, all of which adds to the problem. When this completely unforeseen condition was discovered, ordinary timbering was the only answer, because of the impossibility of obtaining adequate steel supplies in time due to war shortages. Then by the time steel supplies were obtained and a practice developed, we found ourselves with between four and five miles of main haulage drifts and plat openings of a permanent nature, all of which were supported by hardwood timber, some of which had already rotted to an unsafe degree. Also, our drifting program was adding some two miles per year to the above figure.

F. J. HALLER, Member AIME, is Superintendent, Mather Mine, Cleveland-Cliffs Iron Co., Ishpeming, Mich.

AIME Columbus Meeting, September 1949.

TP 2840 A. Discussion (2 copies) may be sent to Transactions *AIME before May 31, 1950. Manuscript received Aug. 22, 1949. Revision received Jan. 17, 1950.*

To give all of the details of our experiments, studies, and experiences would fill a large book, but we were satisfied that our organization had done an excellent job, starting from scratch. You can appreciate our surprise when we discovered that we had added very little to an existing record. Attention was called to a footnote in a copy of "Carnegie Pocket Companion" year 1923. The note read: "Full information as to uses of H-Beams is given in pamphlet entitled 'Steel Mine Timbers.'" A search of our Engineering Library brought to light a pamphlet printed in 1911 showing extensive use of steel for this purpose. Except for welding in place of riveting, we have added very little to the designs in that book. We have, however, developed an experience which proves, at least to our own satisfaction, that steel mine supports pay off under the proper conditions. We are now convinced, in spite of the relatively high initial cost of materials, that steel sets, because of comparatively low installation cost, are actually cheaper than either treated or untreated wood sets, if they are properly designed and adapted.

Several Sections in Use: Convenience guided our first experiments in steel sets, since for a number of years, we had used the 4-in. WF, 10 lb per ft section in 10-ft lengths as safety forepoles along with wood forepoles. This section replaced the 40-lb rail which had formerly been used for the purpose. Having the 4-in. WF sections on hand, it was natural that our first experiments in steel sets involved their use. Later, the rolling of the 10-lb section was discontinued in favor of the much stronger 13-lb section, which is now standard for both legs and caps in areas of no particular weight, but which, none the less, require permanent support.

Our experiments involved the use of the 10- and 13-lb sections as legs and 6- and 8-in. I beams, 16 and 23 lb per ft, respectively, as caps. Experience soon indicated that, if the ground were heavy enough, and the opening wide enough to require a cap stronger than 13-lb, the 8-in, 23-lb should be used, eliminating the 6-in. I beam. We now know that the 8-in. WF section of the same weight, which we propose to use in the future, is considerably stronger for the amount of money involved.

Comparative Strengths: Tables I to IV giving strengths of several species of timbers as compared with the structural sections that we now use have been prepared in accordance with accepted practice. Experience indicates that failure due to compression perpendicular to the grain of the wood does not destroy the effectiveness of a wood set frequently enough to be a practical factor. Therefore, no table of these values is included.

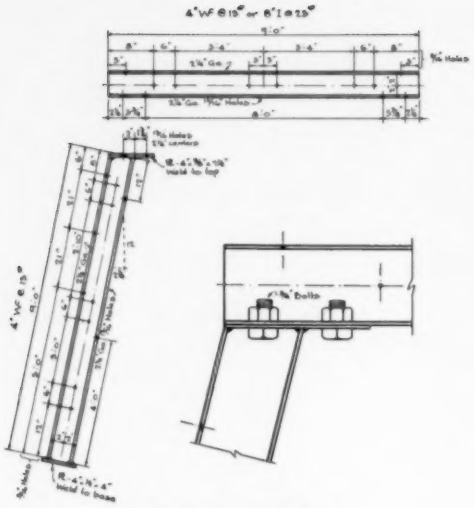

Fig. 1—Diagram of steel sets.

Table I. Timber Caps, Uniform Load, 8-ft Span

Ultimate *Breaking* and *Safe* Working Loads in Accordance with the Stresses Recommended by the U.S. Forest Products Laboratory

Size Diam In.	Maple		Tamarack		Spruce	
	Ultimate, Lb	Safe, Lb	Ultimate, Lb	Safe, Lb	Ultimate, Lb	Safe, Lb
12	134,000	24,100	102,000	20,500	81,000	18,400
13	169,000	30,600	129,000	26,000	103,000	23,400
14	220,000	38,200	162,000	32,800	128,000	29,200
15	259,000	46,900	198,000	40,000	157,000	35,900
16	314,000	56,900	240,000	48,500	191,000	43,500
17	378,000	68,500	290,000	58,300	229,000	52,300
18	448,000	81,000	343,000	69,100	272,000	62,000
	Ultimate and Safe Working Loads Due to *Horizontal Shear* in Accordance with the Shearing Stresses Recommended by the U. S. Forest Products Laboratory					
12	248,000	24,700	146,000	17,900	129,000	16,200
13	291,000	28,800	171,000	20,900	151,000	18,900
14	337,000	33,600	198,000	24,200	175,000	22,000
15	387,000	38,400	228,000	27,900	202,000	25,200
16	441,000	43,800	260,000	31,600	230,000	28,700
17	496,000	49,300	292,000	35,700	258,000	32,300
18	558,000	55,400	328,000	40,200	290,000	36,400

In table I, it will be noted that in practically all cases the shearing stresses govern the safe working loads.

Table II. Steel Caps, Uniform Load, 8-ft Span

Ultimate Loads (Elastic Limit) and Safe Loads (A.I.S.C. Spec.)

Size	Ultimate	Safe
4 in. WF 10 lb[a]	12,100 lb	7,000 lb
4 in. WF 13 lb	15,900	9,100
8 in I 23 lb	46,700	26,700

[a] No longer rolled.

At one time we used a 6-in. I beam, 16 lb per ft, with an ultimate load of 29,400 lb and a safe load of 16,800 lb. We no longer use this section, changing from the 4-in. WF, 13-lb to the 8-in. I beam, 23-lb where heavy conditions are encountered or where spans exceed 8 ft.

The first use to which the tables can be put is to rule out the breaking loads in favor of the lower horizontal shear loads, with the exception of the 12-in. maple cap, where the breaking load governs. In all other cases, the ratio of span and diameter is such that horizontal shear governs the safe load. The 4-in, 13-lb WF section as a cap initially has approximately 40 pct of the safe load strength of a 12-in. diam green hard maple, 45 pct of tamarack and 50 pct of the strength of spruce. These strength ratios become less favorable for steel as the diameter of the timber increases. However, within a very short time, wood loses the advantage in strength due to aging. The 8-in, 23-lb I beam, when used as a cap, compares very favorably with the 13-in. diam maple, tamarack and spruce, the last of which it exceeds by some 3000 lb. Inasmuch as judgment of ground is the ruling factor, we quite frequently use the 8-in. section where heavy conditions are expected. Where light conditions are expected and weight is encountered later, additional sets of the 13-lb section are installed.

Table III. Timber Columns, 9 ft 0 in. Long

Ultimate Breaking and Safe Loads in Accordance with the Stresses Recommended by the U. S. Forest Products Laboratory

Size Diam In.	Maple		Tamarack		Spruce	
	Ultimate, Lb	Safe, Lb	Ultimate, Lb	Safe, Lb	Ultimate, Lb	Safe, Lb
12	455,000	164,000	393,000	136,000	300,000	110,000
13	532,000	192,000	461,000	159,000	354,000	129,000
14	617,000	223,000	534,000	184,000	411,000	150,000
15	710,000	256,000	614,000	212,000	472,000	172,000
16	806,000	292,000	699,000	241,000	537,000	196,000
17	911,000	330,000	789,000	272,000	606,000	221,000
18	1,043,000	369,000	885,000	306,000	680,000	248,000

Table IV. Steel Columns, 9 ft 0 in. Long

Ultimate Loads (Elastic Limit) and Safe Loads (A.I.S.C. Form.)

Size	Ultimate	Safe
4 in. WF 10 lb[a]	70,000 lb	32,300 lb
4 in. WF 13 lb	94,700	42,800
8 in. I 23 lb[b]	111,200	56,600

[a] No longer rolled.
[b] Up to now we have had no occasion to use this section as a column.

In comparing steel with wood for timber columns or legs, once again the original strength is definitely in favor of the wood. The 4-in., 13-lb section, with its safe load of 42,800 lb, has only 26 pct of the strength of a 12-in. green maple, 31 pct of the strength of tamarack and 39 pct of the strength of

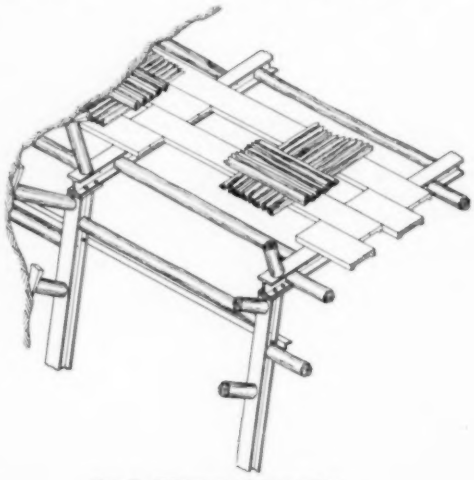

Fig. 2—Diagram of spragging.

spruce. Once again, the rapid loss in strength of wood makes the use of steel economical. Further, the strength of steel is predictable and the strength of wood definitely unpredictable. The tables are based on certain specific moisture contents of wood which are subject to considerable variation.

Another factor which up to now has not been reduced to formula, definitely favors the steel supports. The fact that the cap is securely bolted to the top of each leg with four ¾-in. bolts greatly increases the strength of the set as a whole when compared with wood sets, which cannot be securely fastened together. A combination of top and side weight quite frequently destroys the effect of a wood set, while steel of a lighter theoretical strength under the same conditions does an adequate job of support.

Up to now, we have had no occasion to use anything stronger than the 4-in., 13-lb section as legs, except in areas of extreme weight, where an entirely different section is used. This will be discussed later.

We have prepared for this discussion a number of illustrations showing the method of shop fabrication and underground installation. As mentioned previously, we have added little except experience and the use of welding in the place of riveting.

Fig. 1 is a detailed shop drawing. Accuracy is obtained by performing all drilling and welding in a jig. Reasonable accuracy is necessary, since it is important that all four ¾-in. bolts be used at both ends of the cap for proper rigidity.

Proper spragging is important if steel is to be installed as the heading is advanced (fig. 2). Conventional wood lagging and blocking can be used above permanent carrying poles where fireproofing is not important. "Minecrete" backpoles are 2¼ x 8 in. and carry a uniform load of approximately 2700 lb over a 5-ft span (figs. 3 and 4).

Fig. 5 illustrates permanent, fireproofed carrying poles and blocking versus temporary wood poles, lagging and blocking. New, stronger Bethlehem "hat-section" carrying poles now are in use in heavier areas.

Proper Application Important: To enlarge on my qualifying remarks regarding the proper design and application of steel sets: there are places where we cannot afford to use steel and also there are numerous places where we cannot afford *not* to use steel sets.

Fig. 3—Treatment of permanent plat opening close-covered with "Minecrete" back-poles and blocked with cinder blocks where fireproofing is essential.

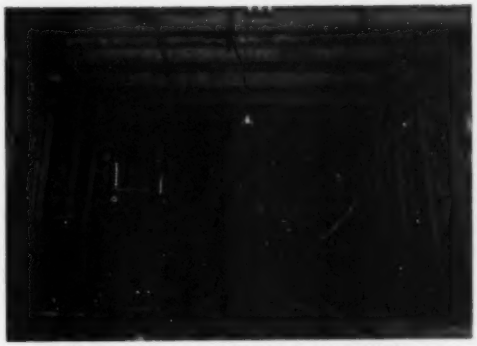

Fig. 4—18-ft span adequately supported.
Permanent fireproof side lagging was accomplished with "Minecrete" poles.

Normally, our mining openings, such as sublevel transfer drifts and exploration drifts in ore, are not intended to last beyond the normal 3 to 4 year life of untreated timber. Also, we can usually expect weight far in excess of the strength of the lighter, cheaper steel sets. Therefore, we use large size green timber, quite frequently reinforced with additional lining sets between the originals, in all cases where judgment indicates that the weight will be great and the length of life reasonably short. In general, our main level haulage drifts in ore fall into the same category, with a number of exceptions, where steel sets close together have done a satisfactory job.

The openings where steel supports are an economic necessity are, for the most part, permanent main level openings around the shaft stations and permanent main level haulage drifts in the footwall rock. There have been no instances in the several miles of main level haulage drifts and permanent openings where the use of steel did not prove economic. We have also found that the use of steel is a must in any areas where fireproofing is indicated, particularly in the vicinity of the shaft, in isolated airways, and in the vicinity of powder magazines. We have also found that steel sets are most desirable in areas which were originally supported with wood sets, where weight has not proved excessive and where a change in plans requires support beyond the life of the timber. This, of course, is true in our main level drifts driven during the time when adequate steel supplies were not obtainable.

Smaller Opening Required: Originally, we found it very difficult to control blasting in a main level rock heading to permit the installation of steel sets as the drift was advanced. Constantly changing ground and an unavoidable turnover in personnel made it impractical to install these sets because of the large percentage of legs and caps which were damaged in blasting to the extent that they had to be replaced. During the past year, however, we have successfully driven more than a mile of main level rock drift with steel sets at 5 ft centers without any appreciable damage to the supports. This is a large factor, since the opening required for 12-in. diam timber sets is some 25 to 30 pct larger than the opening required for steel.

Comparative Costs: The following presents average costs, including freight and handling, for prefabricating steel sets as done in our company's gen-

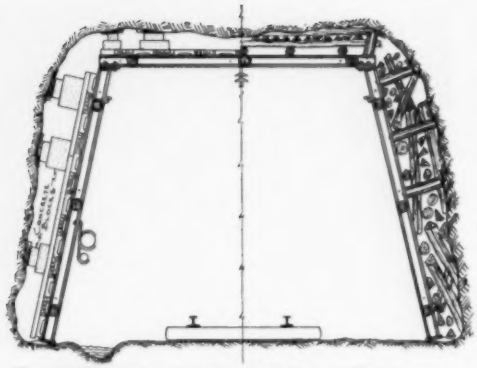

Fig. 5—Diagram of permanent fireproofed carrying poles and blocking vs. temporary wood poles, lagging, and blocking.

eral shops, as compared with the cost for wood sets. I do not propose to discuss the comparative costs or advantages of untreated wood versus treated wood, since in practically every instance where the life of untreated wood is to be exceeded, steel sets are indicated. Our average cost for a prefabricated steel set, including shop costs, freight and handling to the shaft collar, is $20 for a three piece 9-ft set of the 4-in., 13-lb WF section. This compares with average costs for hard maple, including freight, unloading, and framing, of $8.50 for 12-in. diam, $10 for 14-in., and $10.50 for 16-in. I do not have comparative costs for treated timber of the same grade and diameter, but understand that, in general, the cost of peeling, seasoning, and treating about equals the original purchase cost. If the steel set is to be used as an original installation in a rock drift, the first saving is the above-mentioned 25 to 30 pct decrease in the volume of rock excavated. To reduce this saving to actual money would be very difficult, but it is obvious, at least in our case, that it is considerable. The time for installation is approximately one-half of that for wood sets, with a cash saving which varies with the wage scales and skill of the men, but which averages $4 per set. The best comparison between installation costs can be obtained where a main level haulage drift is being repaired, with the wood sets being removed and new steel sets being installed. We have several crews of repairmen which

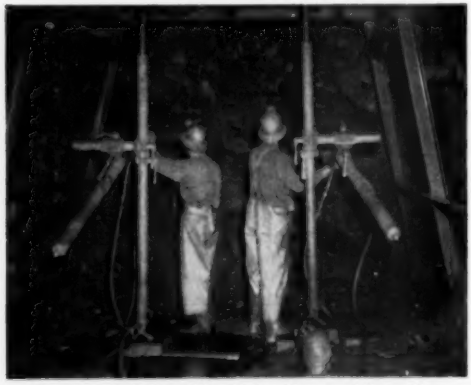

Fig. 6—Typical sublevel development in rock.
Wood carrying poles may or may not be replaced with permanent "Minecrete" poles and Bethlehem "hat-section", depending on anticipated length of life.

give us an average of $10 per set installation costs. This includes a complete job with the new steel set installed, the old wood set removed from the drift, and the back completely covered and blocked. Under these same conditions, with the haulage trains going back and forth, with the necessity for removing the air lines, water lines and trolley wires, it is practically impossible to conduct this repair work except on weekends at penalty rates without seriously hampering the tramming operations. The small 4-in. section permits this work to be done at straight time rates, since the legs can be slipped into place behind the air and water lines, and the caps installed without removing the trolley wire. Under these conditions, the installation cost of a 12-in. wood set exceeds $20 with the same work completed. Obviously, a $20 set which can be installed for $10 costs about the same as an $8 or $9 set with $20 for installation. The steel set, if put in an area not beyond its capacity, will last forever. The wood set untreated will last 4 years and, quoting other authors, "if treated will last not more than 15, with an average of less than 10 years."

In the repairing of important transfer drifts under large stopes or block caves, the same advantages of the steel sets hold true. So many more can be installed by the same number of men under heavy conditions that, in a great many cases, it is far cheaper to repair a wood drift with steel than to repair it with more wood.

Transportation and Handling Reduced: The transportation of timber and timbering supplies is another important cost item which is difficult to reduce to dollars and cents. In the case of the 9-ft, 4-in. steel sets, the average load on a timber truck is 18 sets, or 54 pieces, which compares with 7 pieces of 12-in. diam hardwood. In other words, transportation and handling cost from the timber yard or shop to the working place is approximately ⅛ the cost of wood. This is a very important item in our operation, where labor accounts for approximately 60 pct of the cost of the product.

As previously mentioned, the average mining drift in ore has too short a life to warrant steel in the original installation and is usually too heavy to be supported by the lighter, cheaper sections. However, there are a great many exceptions to this general rule in the 4000 ft of combined development openings driven at this property each month over the past several years. There are innumerable instances in stope and block cave transfer drifts where steel is absolutely essential, particularly in supporting the area in the immediate vicinity of mills and draw raises. Numerous methods are used and in some instances we have gone to a 6½ x 12-in. WF, 28-lb section for both legs and caps. This section, which is approximately 2¼ times as strong as the 8-in. standard I beam, serves admirably in areas where the side pressure is equal to, if not greater than, the weight on the cap.

In a number of instances, we have driven semi-permanent top timber carloading transfer drifts in rock and contact material. Invariably, since the excavated material is not salable, we are desirous of cutting down the amount excavated. The use of steel supports in these openings is mandatory because of the indicated length of life, the decrease in the excavation, and the absence of great weight as a general rule. However, in a number of these drifts where great weight was later encountered, additional lining sets have done an adequate job.

Recent Developments in the Manufacture of

Lightweight Aggregates

by John E. Conley

and

John A. Ruppert

Shortages of cinders, largely resulting from industrial plants converting to fuel-oil, together with an enhanced building activity, have seriously affected the cinder block and lightweight concrete manufacturers. In the search for other sources of lightweight aggregate, various clays, shales, slates and natural occurring lightweight rocks such as pumice, scoria, volcanic ash and the like have received special attention. Methods and equipment for producing suitable lightweight aggregates have been considered. In addition to an expanded use of the rotary kiln, the traveling-grate and continuous-sintering machines are being installed in many localities.

LIGHTWEIGHT aggregates have been in use for many years in the United States but are now receiving more and more attention by manufacturers and users of concrete shapes. These shapes comprise building blocks, special forms, such as door sills, lintels, floor slabs, and beams. Lightweight aggregate also is adapted for monolithic construction. The increased building activity since termination of the war, together with the widespread shortage of cinders resulting from a general trend toward the use of oil and pulverized-coal fuels, has aggravated the situation.

In some localities, new producers of lightweight aggregates have been stimulated to increased efforts to supply the demand. In addition, various establishments of the United States Government have recently given attention to this problem. The National Bureau of Standards, in the Commerce Department, and the Bureau of Reclamation, in the Interior Department, have completed a series of tests on concretes made from some of the available commercial aggregates in order to evaluate their qualities. The tests have been made to focus attention on the desirable properties of good aggregates. This work was done at the request of the Technical Office of the National Housing Agency and Housing and Home Finance Agency. Some time previously, the former office had conducted a survey of the locations where lightweight aggregates are produced.[1] A study of this information discloses that the producing centers are widespread and that certain areas, notably the region east of the Mississippi River and especially the southeastern states in the sections along the Atlantic Coast, have only a limited number of such plants. The situation has been alleviated to some extent with new plants having been erected near Richmond, Va., and Lansford, Pa. Experiment Stations of the Federal Bureau of Mines at Tuscaloosa, Ala., Norris, Tenn., and College Park, Md., have investigated types and sources of potential raw materials such as clays, shales, and slates, as well as methods and procedures for utilizing these materials for their conversion into aggregates, and have published the data as a Bureau of Mines report.[2]

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Structural Uses of Lightweight Aggregates: Production of masonry blocks in the United States has averaged close to a billion units a year for the past three years, and of these it is estimated that more than 40 pct consisted of the lightweight type. It is anticipated that the future will witness an increase in this percentage in view of the superior insulating properties, decrease in dead weight, and ease of handling, transporting, and laying units of this type.

Lightweight aggregates are used extensively in multistoried buildings in which the weight of the walls and floors is carried by a structural steel framework. The substitution of a lightweight concrete for normal concrete in a building of this type has brought about in many instances some or all of the following advantages:

1. A reduction in the amount of structural and reinforcing steel needed.
2. Better heat and sound insulation in walls and floors.
3. Lower costs for the construction of concrete forms.

The advantages gained by using lightweight concrete is illustrated by changes in the original design just before construction of a 14-story annex to a large department store in Cleveland, Ohio.[3] The initial plans of the building contemplated three mezzanine floors using concrete of crushed stone and sand. Before ground was broken, however, the owners decided in favor of seven additional mezzanines, or a total of ten. Foundation limitations made the use of lightweight concrete mandatory, and tests were carried out by a private laboratory to arrive at proportions that would consistently yield concrete of the required strength and workability. Specifications were set up that readily met the 3000-lb compressive-strength requirement. With no sacrifice in strength, it was possible in this instance to effect approximately an 8000 ton reduction in weight to be supported by the steel columns.

By utilizing lightweight concrete, there also exist many instances in the United States where it has been feasible to add a greater number of stories to existing buildings than would otherwise have been possible with the conventional sand-gravel concrete.

During World War II, lightweight aggregates were used in making concrete ships and barges. Considerable savings were effected by this method of converting dead weight to cargo capacity.

The use of a lightweight aggregate in place of gravel or crushed rock in the concrete roadway of the San Francisco-Oakland Bay Bridge, completed in 1936, is said to have effected a saving of three million dollars because of the decreased quantity of structural steel necessary.

In recent years, there has been an increased use of superlight aggregates in concretes that are characterized by their excellent heat-insulating values. Because of their extremely low weight, concretes made from this type of aggregate cannot be expected to have high strength characteristics. Where strength is essential, more cement must be used, and an increase in weight results. However, strengths are adequate for such uses as fireproofing for steel beams, non-load-bearing subfloors and insulating sections of walls and roofs. Farmers in the colder areas of the United States have discovered that a subflooring of insulating concrete will prevent condensation in barns, thereby keeping the floor dry and warm and contributing measurably to the health of the animals housed therein.[4] Another advantage of concretes of this type is that they usually can be nailed or sawed.

Types of Aggregates: The aggregates finding commercial acceptance today in the United States might be roughly classified into two groups. The first comprises the extremely light materials, such as exfoliated vermiculite and expanded perlite, weighing 3 to 25 lb per cu ft and utilized in concretes where high-insulating characteristics are desirable. These materials usually yield concretes having 28-day compressive strengths in the range of 100 to 1000 psi. Because of their extreme bulk, aggregates of this nature seldom are transported long distances in their expanded forms but are shipped in the raw state and processed in the areas where they are consumed.

Aggregates of the second group include those weighing 30 to 70 lb per cu ft and capable of yielding concrete with 28-day compressive strengths of 1000 to 5000 psi or more in some instances. These medium- and high-strength aggregates might be further classified as to methods of manufacture as follows:

1. Naturally occurring aggregates.
2. Expanded or foamed slags.
3. Bloated aggregates produced in rotary kilns.
   a. Crushed to size after firing.
   b. Crushed or extruded to approximate size before furnacing and then fired without agglomeration.
4. Sintered aggregates.

Methods for producing aggregates of both groups will be discussed briefly with particular emphasis on history and recent developments.

Vermiculite: Vermiculite is a hydrated magnesium aluminum silicate that possesses the extraordinary characteristic of expanding from 6 to 20 times its original volume upon being heated. In the United States, it is mined in Montana, Wyoming, Colorado, South Carolina, and North Carolina. The expansion or exfoliation appears to be due to the generation of steam from water that occurs between cleavage planes of the mineral, and the processed material may have a bulk density as low as 6 lb per cu ft. Exfoliated vermiculite has long been used as a loose-fill insulation but has been employed only in recent years as an aggregate in concrete and plaster. Concrete made from this material usually weighs 25 to 50 lb per cu ft, depending on the quantity of cement used in the mix.

Perlite: A relative newcomer to the lightweight aggregate field, perlite is an acidic lava containing 2 to 5 pct water and is capable of expanding with disruptive violence 6 to 10 times its original size when subjected to rapid heating. Deposits are located only in the western areas of the United States, but processing plants are being erected in numerous cities, one as far east as Carnegie, near Pittsburgh, Pa.[5] With some types of furnaces, it is possible to expand suitable perlites to a product weighing as little as 3 lb per cu ft; however, material this light, because of its very friable nature, is generally used as a loose-fill insulation rather than as an aggregate. Perlite that has been expanded to a lesser extent is used to make concretes weighing 35 to 60 lb per cu ft depending upon cement content.

Naturally Occurring Aggregates: These include pumice, pumicite, volcanic ash, scoria, and diatomite. Pumice, the most widely used of these materials at present, is of volcanic origin and occurs only in the western regions of the United States. Considerable activity has taken place during the last three years, both in the development of new deposits and the study of techniques of mixing pumice concrete.[6, 7] Pumice is mined in California, New Mexico, Oregon, Utah, Idaho, and Washington. Some of the more progressive producers have set up screening as well as crushing equipment, so that the product can be shipped separately in two or more sizes. The consumer is thereby able to choose the proper ratio of coarse to fine aggregate for each batch of concrete. Ungraded crusher-run pumice has the disadvantage of tending to segregate during shipment so that there is a danger of successive batches of concrete being nonuniform.

Compressive strengths of 2000 to 3000 psi are claimed for properly mixed concrete containing pumice from one source. Higher values are possible when sand is incorporated.[6] It must be kept in mind that deposits of pumice occur in only five states and even within a single deposit it varies considerably in strength and other properties. Therefore, the results of strength tests made on concrete from a specific pumice may not be valid for pumice mined elsewhere.

Pumice weighs approximately 30 lb per cu ft for the coarse sizes and 45 lb per cu ft for fines. One producer has quoted a price range from $2.07 to $2.40 per cu yd at his plant, depending upon the quantity ordered.

One disadvantage of pumice is its relatively high water absorption, although some authorities claim that presoaking and proper handling will overcome this difficulty.

Pumice concrete has been widely used in California, especially in the construction of various buildings in Los Angeles and vicinity. Those of the Pacific Telephone and Telegraph, Prudential Insurance, and General Petroleum companies are examples. In the latter instance, it has been reported that an additional expenditure of $61,000 for pumice concrete decreased the structural steel cost by $180,000. Better heat and sound insulation as well as increased resistance to earthquake damage are other possible advantages.

Expanded Slags: Expanded or foamed slags, under various trade names, have been marketed for a number of years in the United States and even longer in Europe. Usually, these products are made by causing a controlled quantity of water to come in contact with a molten blast-furnace slag, the resulting steam acting as a vesiculating agent. The amount of water used is critical, as an excess will produce a granulated slag which is fragile and unsuited for use as an aggregate whereas an insufficient quantity will bring about the formation of a heavy product.

The simplest method of manufacturing a foamed slag consists in allowing the molten material to run into a pit shaped as a truncated cone with a depth of approximately 15 ft and a width about 20 ft at the top and 15 ft at the bottom. Prior to the introduction of the slag, a quantity of water amounting to roughly one-third of the weight of the molten material is allowed to run into the pit. The expanded mass is subsequently removed by a clamshell bucket and transported to a crusher that produces the desired sizes. In a variation of this method water is sprayed on the stream of molten slag as it is poured into the pit.

One machine that has found considerable use allows the slag stream to fall upon a series of paddles that rotate in a horizontal plane about a central drive shaft. The paddles break the stream into small particles which are deflected to a second and third group of rotating paddles by means of buffer plates. A controlled stream of water plays upon the moving portions of the machine and the buffer plates. The violent agitation coupled with the action of the water turning to steam causes the slag to expand during its course through the device. The product is crushed and graded in the usual fashion.

Although the machine-foamed aggregates are satisfactory for many uses, apparatus-maintenance costs are appreciable, inasmuch as parts exposed to molten material at 2550°F tend to burn up and must be replaced. In an effort to reduce production costs and still produce an acceptable foamed slag, a British organization is attempting to introduce into the United States a recently developed, modified pit process which is said to have attained a considerable degree of success in England. Pilot-plant tests are now being conducted on American slags of the steel mills in the Pittsburgh area. The new method differs from the old pit process in that only a very small quantity of water is introduced into the pit before pouring the slag therein. Immediately after pouring, a considerably larger amount of water is ejected upwards by means of nozzles embedded in the floor of the pit. The thin layer of water on the floor initiates the foaming of the slag and that fed through the jets at the bottom of the pit completes the operation. The degree of foaming can be controlled by both the total amount and rate at which the water is injected up in the molten mass. It is alleged that 10 to 20 tons of slag can be foamed in less than 2 min. Pits of this type can be built of concrete and located outdoors. If desired, they can be made shallow and with sloping sides so that the foamed material can be removed immediately after processing by means of a mechanical pusher, caterpillar tractor, or bulldozer. The sponsors claim that the aggregate made by this patented process[7] is of a quality comparable to the machine-made type and can be manufactured at a much lower cost. The construction of the pit used to foam the slags is illustrated in fig. 1.

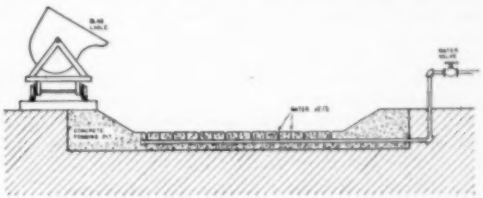

Fig. 1—Schematic drawing of section through concrete slag-foaming pit.

Weights of foamed slags made in the United States are approximately 45 lb per cu ft for the coarse sizes and 55 lb per cu ft for the fine. Prices averaged $1.88 per cu yd f.o.b. plant in 1948. Because of the low price of foamed slag as compared with other manufactured aggregates, this product finds considerable favor with concrete block manufacturers. Most of the output is used for this purpose.

Bloated Aggregates: There are many localities where a demand for lightweight aggregates exists but which are remote from natural deposits or blast-furnace installations producing foamed slags. Therefore, because of high freight rates in relation to the value of the product, these areas are obliged to rely on aggregates produced by treating certain argillaceous materials, such as clays, shales, or slates. These are heated to bring about expansion or bloating. Materials of this type were studied by the Federal Bureau of Mines.

In order for the bloating of an argillaceous material to occur, two conditions must be fulfilled. First the material must become soft and somewhat plastic under the influence of heat, and second, gases must be evolved simultaneously with the existence of the pyroplastic condition. The resultant product is a glassy mass containing numerous small blebs or vesicles. Uniform cells of small size insure maximum aggregate strength and, when separated by cell walls and with a minimum of connecting pores, favor minimum water absorption.

Up to the present time, virtually all commercial production of bloated aggregates has been in the rotary kiln by means of one of two methods. The first method consists of feeding the raw material in the form of fairly coarse sizes, allowing a certain degree of fusion or agglomeration to take place while firing and afterward crushing the expanded product to sizes suitable for aggregate use. The second method involves either crushing or extruding the feed to a size somewhat under that desired in the final product and bloating the material in the kiln as separate particles. Agglomeration is avoided, thereby eliminating final crushing.

Bloated aggregates have been manufactured sporadically and on an experimental basis since 1913, but commercial production did not get underway until 1920 at a plant in Kansas City, Mo. Standard practice for manufacturing aggregate by the first mentioned method, in which the aggregate is crushed after firing, consists in heating shale or other argillaceous material crushed to minus 1½ or 2 in. size in a 6x60 ft kiln using oil, coal, or natural gas as fuel. The material is discharged as clinker which, after cooling, is crushed and graded, producing an aggregate of moderately high strength suitable for use in monolithic concrete structures. Production of a 6x60 ft rotary kiln for a 24 hr day is of the order of 100 to 150 cu yd, generally closer to the former figure and fuel consumption may be 10 to 16 gal of oil or 250 to 350 lb of coal per cubic yard of aggregate.

An 8½x70 ft kiln has been installed recently at a plant near Portland, Ore.[9] With the exception of two similar kilns 175 ft long, which are in operation at Ventura, Calif., this is believed to be the largest-diameter rotary kiln now in operation in the United States for processing a lightweight aggregate. It is rotated at a speed of 2 rpm and pitched at a slope of ⅜ in. per ft. Feed is a shale crushed to minus 2 in., and production is alleged to be 268 cu yd per 24 hr day which is far in excess of the reported output of any other single kiln making lightweight aggregate. For instance, the same diameter but longer kilns of the Rocklite plant at Ventura, Calif., are claimed to produce only 175 cu yd of aggregate per day. However, a part of the difference might result from the type of raw material used. Whether or not this will set a trend in kiln sizes for this industry remains to be seen.

Bulk densities of this variety of aggregate are approximately 40 lb per cu ft for the coarse sizes and 60 lb per cu ft for the fine. The present price ranges upward from $4.00 per cu yd f.o.b. plant.

Among the problems that beset the operators of a plant using a rotary kiln to produce an expanded aggregate are those of preventing the material within the kiln from sticking to the lining or agglomerating into balls so large as to make their removal difficult. It is unusual to find a natural argillaceous material that will bloat and yield a good lightweight aggregate without showing some tendency to adhere either to itself or the kiln lining or both. Although some materials will bloat over a wider temperature range before clinkering or sticking than others, the problem is ever present. The trouble is largely unavoidable, because the charge must be relatively soft and plastic and gas must be evolving before vesiculation can take place. Ring formation within the kiln may be aggravated by the deposition of ash from powdered coal, in many localities the most economical type of fuel. In an effort to minimize excessive agglomeration and sticking, a number of installations are equipped with two burners. One burner is directed along the axis of the kiln and heats the material nearly to the bloating temperature while the other plays directly upon the charge within a few feet of the discharge point. Thus, the second burner actually raises the material up to bloating temperature. The successful firing of a material, such as a raw shale, in a rotary kiln requires a high degree of skill on the part of the operator, who must keep the temperature within a relatively narrow range. Variation of temperature will produce a heavy, underbloated aggregate if firing temperatures are too low or will fill up the kiln with a sticky, gummy mass presenting problems of removal if the temperature is allowed to get too high.

Various schemes have been suggested to avoid processing troubles caused by the adhesive tendencies of bloating materials as well as to produce an aggregate consisting of discrete, well-rounded particles, each having a uniformly cellular interior sealed off by a hard impervious surface. Firing can be accomplished without causing the particles to agglomerate on their way through the kiln. Subsequent crushing thereby will be rendered unnecessary. Of these proposed methods, most have been abandoned because of high production costs.

Considerable work was done on a patented process[10, 11] in southern California, nearly 20 years ago whereby clay was pugged with water and heavy fuel oil. The latter functioned as a bloating agent. The plastic mass was extruded through a multiple die, and the "macaroni" was cut into short lengths to make pellets. Before firing, the particles were coated with a powdered refractory material having a higher fusing point than the clay. Upon traveling through the kiln each particle was able to maintain its separate identity and yielded spherical or rounded shapes with sealed outer surfaces.

At present, a producer of lightweight aggregates in Strawn, Texas, is known to be making a similarly pelletized product, although only on a limited scale. During World War II, a special pelletized lightweight aggregate was made in a plant at Ellenton, Fla. Three grades of aggregate were produced, as follows: —⅝ +⅜ in., —⅜ +¼ in., and —¼ in. These materials were shipped to Tampa, Fla., for use in constructing concrete ships. However, it is believed that production costs in the war-built plant were too high to justify peacetime operation.

Another process[12] along similar lines has been attempted in which a crushed diatomaceous shale has been fed to a kiln simultaneously with a small quantity of powdered limestone or dolomite. Upon entering the hot zone, the surfaces of shale fragments become sticky and pick up a coating of lime which prevents agglomeration. A portion of lime also may react with the silica present in the shale, thereby forming a shell of calcium silicate on the

surface of each particle. This process is believed to have had a commercial trial.

Another lightweight-aggregate plant which functioned during World War II to supply a high-grade lightweight aggregate for concrete-ship construction was that at Santa Maria, Calif. Their product is the commercial aggregate sold on the market as "airox." The raw material used to produce airox is a diatomaceous earth with some clay and has a high-oil content that can be reclaimed by heating or distillation. To expel the excess oil and obtain a satisfactory lightweight aggregate, the crushed raw material was treated during the war in two rotary kilns operated in series. The hydrocarbon content of the material was burned out in the first kiln, while the sintering or incipient fusion necessary to produce a strong aggregate was accomplished in the second kiln. The costs of this method of handling the aggregate were high and warranted only on a wartime basis.

Since the war, the kiln treatment has been discontinued and the equipment dismantled and sold. However, by merely placing the suitably crushed diatomite in heaps or windrows and initiating combustion of the oil contained in the material by lighting with a torch, an acceptable lightweight aggregate can still be produced. Control of the product is obtained by the method of preparing the piles or heaps for burning and by watching the size gradation. Actual burning requires up to 48 hr, depending on the handling technique and results desired. It is claimed that the finished product can be placed in the finished-product bins at the plant on a basis competitive with crushed pumice.

Fig. 2—Large lightweight aggregate plant constructed to treat coal wastes in Pennsylvania anthracite region.

The operators of the plant located at Ventura, Calif., near Los Angeles, have been fortunate in discovering a deposit of soft shale that appears to be unique insofar as its ability to undergo bloating in a rotary kiln without sticking or agglomerating is concerned. The raw shale is crushed and screened to three sizes each of which is fired separately. The material, in traveling through the hot zone, becomes slightly sticky, but the particles maintain their individuality upon being discharged from the kiln and are not in an agglomerated condition. Furnacing this material requires no prior processing other than crushing and grading to size, and it is believed that this installation is the only one today in the United States in which a raw, untreated shale is bloated in a rotary kiln without having to be crushed after firing. The product consists of well-rounded particles, each of which has a cellular interior sealed off by a smooth outer skin. This is true of the fine sizes, as well as the coarse. As a result, an aggregate is produced having many desirable characteristics, such as high strength, low-water absorption, and good workability. Bulk densities of this aggregate are about 40 lb per cu ft for the coarse and 50 lb per cu ft for the fine sizes.

Sintered Aggregates: Excellent aggregates exhibiting the qualities of high strength, light weight, good workability, and low-water absorption have been made by the established rotary-kiln processes. The relatively high-production cost of aggregates of this type prevents them from finding wide acceptance by the manufacturers of masonry units, especially since certain characteristics, as for example workability, are not too important for this use. Therefore, at present, more efficient types of furnaces other than the rotary kiln are needed to produce medium-quality aggregates for utilization in concrete blocks.

Development has proceeded in the direction of sintering machines in which the raw material and a solid fuel are placed on grates provided with a fan for supplying air for combustion of the added fuel. Machines for this purpose have taken the form of endless-chain, tilting or rotating annular grates. It would appear that a sintering machine, if it could be made to operate successfully, might possess definite advantages. These sintering machines could then be used in place of rotary kilns for the manufacture of a medium-grade, low-cost aggregate suitable for ordinary masonry blocks. Apparent advantages are:

1. The use of a low-cost fuel is permissible. A high-ash coal that might cause ring formation when burned in a rotary kiln might be used to better advantage in a sintering machine. Certain shales exist in which the carbon content is high enough to provide part if not all of the fuel requirements.

2. Agglomeration and fusing of the material being fired would present less of a problem. Shales and slates with small ranges between bloating and fusing temperatures could be handled with more success in a sintering machine than in a rotary kiln. Thus a wider variety of raw materials would become available for conversion into lightweight aggregates.

3. For a given aggregate capacity, a sintering machine would require less capital outlay. Kilns and their accessories are relatively expensive and for a 6x60 ft installation output is limited to a maximum of approximately 150 cu yd per 24 hr.

Twenty years ago there were reports in the literature on the experimental use of sintering machines for manufacturing aggregate.[10, 11] An aggregate named "Lytag" was to have been placed on the market in the early thirties, although it never appeared, possibly because of cessation of building activity at that time. Mention is made of a wide variety of materials that could be used to make this aggregate, including clays, shales, slates, schists, and refuse from the coal mining industry. Processing included crushing the raw material to minus 4 mesh, blending with granulated coal and adding a small quantity of water. The mixture was then discharged in a permeable condition on the traveling grate of a machine similar to the Dwight-Lloyd continuous-grate sintering apparatus. After the charge was ignited, the air for combustion was drawn downward by means of a suction fan. It is alleged that the character of the sinter could be varied by ad-

justing the fuel:clay ratio, the quantity of air drawn through the charge, depth of charge, and rate of travel of the grate. Other patents[15, 16, 17, 18] also disclose the use of a continuous chain grate for the manufacture of lightweight aggregates from argillaceous materials.

Several patents[19] were issued in the early thirties to John E. Greenwalt on methods and apparatus for producing lightweight aggregates by means of a stationary sintering pan that could be tilted to discharge the product.

Passing on to postwar times, three new developments have taken place in traveling-grate machines. The first of these, and the only one as yet in continuous commercial operation, is in the anthracite region of Pennsylvania and was specifically constructed for producing lightweight aggregate from coal-mining wastes. Without doubt, this installation is one of the most interesting and most important developments in the industry.

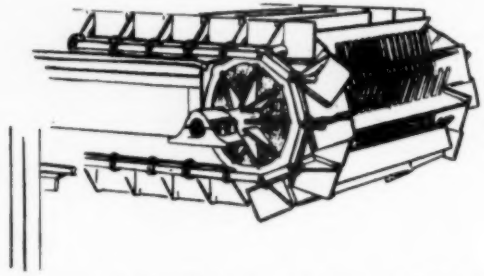

Fig. 3—Rough sketch of traveling grate type of sintering machine illustrating heavy construction of parts.

Fig. 2 is a photograph of the lightweight aggregate plant during construction. Evidence is given in the figure that a large capital investment is involved in plant facilities. The plant is between the towns of Tamaqua and Coaldale in the anthracite coal-mining section of Pennsylvania. Several units are installed at this plant, and a production of 125,000 tons per year is anticipated.[20] The raw material used is a metamorphic carbonaceous shale mined along with anthracite and separated therefrom in a coal-preparation plant above ground. Each bloating furnace consists of an endless traveling grate, said to be the largest ever constructed, operating very much like a chain-grate stoker. An ignition mixture containing anthracite, which is used to promote the bloating process, is fed onto the grate and passes through an ignition chamber. Just beyond the chamber the shale is fed onto the layer of ignited fuel. As it passes through the furnace, combustion of the carbonaceous matter produces a temperature of approximately 2800°F in the bed of material. The particles become softened and are bloated by the entrapment of gases resulting from the burning of the carbonaceous content of the shale itself as well as the release of certain other volatile constituents. Incipient fusion converts the material into a form said to be chemically inert and structurally strong. The product emerges from the furnace in the form of a continuous slab which is reduced to minus 2½ in. size in a primary crusher and then stockpiled. Sizes suitable for use as aggregate are produced in a secondary crushing and grading operation. The final product is said to weigh about 40 lb per cu ft for the coarse and 62 lb per cu ft for the fine sizes.

Another type of endless-grate sintering device is being constructed and sold by a manufacturer of concrete block-making machinery. One unit is now being installed at Marietta, Ohio,[21] and another at Detroit, Mich. The continuous grate is about 50 ft long and 5 ft wide and runs between sprockets approximately 5 ft in diam. An idea of the general type of grate used can be had by referring to the sketch shown in fig. 3. A simple arrangement for using this machine in a lightweight-aggregate plant, together with the other essential units, is shown schematically in fig. 4. This machine differs principally from the one just described for treating coal-mine wastes in that air for combustion is drawn downward through the bed of material on the grate by means of a rectangular suction box connected to a fan driven by a 350 hp motor. Another dissimilarity is that the function of this machine is to form a lightweight material not so much by a bloating action, although this may occur to a degree with some materials, but to agglomerate small granules of clay or shale that go to make up the feed so that the resulting aggregate derives its lightness mainly from the air spaces remaining between the fused particles. Although similar in many respects to the conventional Dwight-Lloyd sintering machine, this device differs by having the grate divided into rectangular compartments by walls transverse of the machine and perpendicular to the grate bars. The grate bars themselves are of a rather ingenious construction being interlaced when the grate is positioned above the suction box but breaking apart when passing over the sprocket at the discharge end and performing simultaneously the functions of discharging the cake and cleaning themselves. In operation, a clay, shale, or other suitable raw material is crushed to minus ⅛ in., sized, and mixed with about 7 pct of finely divided solid fuel such as coal, lignite, or cinders along with enough water to give bulk to the mixture and obtain a satisfactory consistency for sintering. The charge is then passed through a screen that allows the larger particles to fall into one hopper positioned above the traveling grate and the fines to fall into a second hopper behind the first. Material from the first hopper falls onto the surface of the grate and forms a bedding approximately 1 in. thick for a subsequent 8-in. layer of finer material that falls from the second hopper. If desired, material that has previously been sintered may be used for the bedding layer. The charge is then conveyed under an ignition hood where a flame is momentarily applied to the top surface as a blast of air is drawn downward through the grate by means of the fan. The ignited charge continues to advance over the suction box as the burning zone propagates downwards. The rate at which the grate advances is controlled so that the charge is completely sintered by the time it is discharged. The finished cake is crushed to minus 4 in. size, and the minus ¾ in. material resulting from this preliminary crushing

is returned to the machine for further sintering. The remainder is crushed and graded to aggregate size. A production capacity of 30 cu yd of aggregate per hour is claimed for this machine.

A third variety of traveling-grate machine is being marketed in the United States by another manufacturer specializing in concrete-block machinery. Possible future installations may be at Denver, Colo., and Bridgeport, Conn. The prototype of this apparatus is in the New York City area where experimental work has been done with fly ash as a raw material.[22, 23, 24] In appearance, the machine is similar to the Dwight-Lloyd circular-grate type sintering machine first patented in 1909 (U. S. Patent 916,392). The device consists of a series of hinged pans or grates disposed adjacently to one another in an annular configuration, rotatable about its axis. Beneath the grates is positioned a C-shaped suction box that draws combustion-supporting air through the material being sintered. The fly ash, which is said to contain its own fuel supply, is mixed with water in a continuous mixer and flows into the sintering pans, which are continuously rotated. However, before the sintering pans are filled, a 1 in.

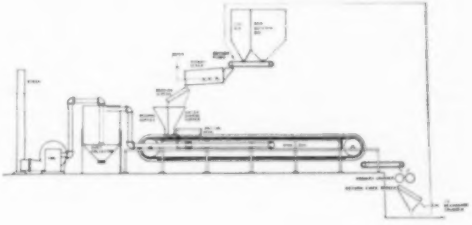

Fig. 4—Schematic arrangement of sintering machine for making lightweight concrete aggregate.

layer of previously sintered and crushed material is spread over the grate to form a permeable bed. About 5 in. of raw material is placed on top of the bedding. After a section is filled, it is ignited by a burner for 1 min, afterward passing over the suction zone for 18 min while combustion is maintained. After sintering is completed, the grate retracts, and the clinker falls out whereupon it is crushed and graded. The grate has by this time rotated through 360° and is ready for refilling and repetition of the cycle. Provision can be made for a preliminary crushing, whereby the fine, insufficiently burned material can be returned for further sintering. It is reported that this machine is adaptable for processing clays, shales, and numerous other materials, as well as fly ash. Solid fuel to the extent of from 4 to 10 pct of the raw material must be added when sintering noncarbon-containing substances. Output for this machine is claimed to be 150 to 200 cu yd of aggregate per 8 hr day.

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Economic Factors in the

Western Phosphate Industry

by Roscoe E. Bell

An economic appraisal of the western phosphate industrial potentials made between 1945 and 1948 revealed possibilities for a considerable expansion in the production of high analysis phosphate fertilizer. The most significant facts are: (1) a rapidly increasing and a much larger potential market which might be reached from the western deposits, (2) the existence of large undeveloped deposits of phosphate rock, (3) potential energy from sulphuric acid, coal, or hydroelectric power in the general vicinity of the deposits. The challenge to the mining and metallurgical engineering profession is in the economical development of phosphate mines and in improving processes for utilization of the western ores and the recovery of byproducts to make the expanded western industry satisfactorily compete with the industry in the East and South in meeting the fertilizer needs of the Midwest and Far West.

BETWEEN 1945 and 1948 the author made studies of the western phosphate industry and its potentialities. These included an appraisal of the opportunities for western development of the industry, studies of production costs, of market potentials, of transportation economics, and of industry location.

This paper will be directed toward pointing up the factors which have an important bearing on the establishment of fertilizer plants in the western area.

Industry Location Factors: There are, of course, an infinite number of possible organizational setups which might require consideration. However, the phosphate industry is simple enough so that studies of the most probable alternatives can be made conveniently. The salient factors which require special consideration in such a study are: (1) the location of the raw materials, (2) the location, size, and character of the present and potential markets, (3) the location of energy sources and materials required for fertilizer processing, (4) the location and capacity of present manufacturing facilities, (5) the cost of assembling the materials and energy needed in manufacturing, and (6) the cost of distributing the manufactured product.

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Raw Material Supplies: Rock phosphate is the raw material for the manufacturing of phosphate fertilizer. In the United States the important known deposits of this material are located in three general areas (fig. 1). The western deposits, comprising about 60 pct of the nation's supply, are in the four-state area—Idaho, Wyoming, Utah, and Montana. The Florida deposits comprise roughly 38 pct of the national supply. The Tennessee deposits, comprising about 2 pct of the national supply, have been the most important source until recent years. The western deposits, which are virtually undeveloped, have not been used to any extent to meet the nation's need for phosphate rock and phosphate fertilizer.

Fertilizer Markets: The large national markets for fertilizer are in the midwestern and eastern United States (fig. 2). The principal far-western markets are along the Pacific Coast, especially in California. Until recent years the western markets have been extremely limited, whereas the markets in the eastern United States have long been established and manufacturing facilities have been provided to meet these needs (fig. 3). In recent years the western markets have expanded very rapidly, continuing to grow throughout the war years and postwar period at a similar rate. The question always arises as to whether this is a temporary war-swollen market which will shrink back to its original size as soon as the present agricultural price levels drop. In order to answer this question consideration must be given to the character of the market (fig. 4). I have considered seventeen western and midwestern states in the western market area where transportation costs from the western deposits are equal to or lower than those from Florida. They include all states west of a line drawn through Chicago, Kansas City and the western boundary of New Mexico. In the prewar period this was a small market totaling only 37,000 tons of P_2O_5 annually. The 1946 total for this same area was approximately 200,000 tons. Considerably more fertilizer would have been used in 1946 had adequate supplies been available.

Estimates of future markets are difficult to make, but we do have some indices that are useful. The

Fig. 1—Major deposits of phosphate rock in the United States.

Source: Mansfield: *Ind. and Eng. Chem.* (1942) 34, 9-12; and Phosphate Fertilizer Consumption, Production Capacity, and Reserves in Western United States. U.S.D.A. Int. Bur. Comm. on Postwar Prog. 1943; U.S. Dept. of the Interior, Bonneville Power Admin., Div. of Ind. and Res. Dev. 1946.

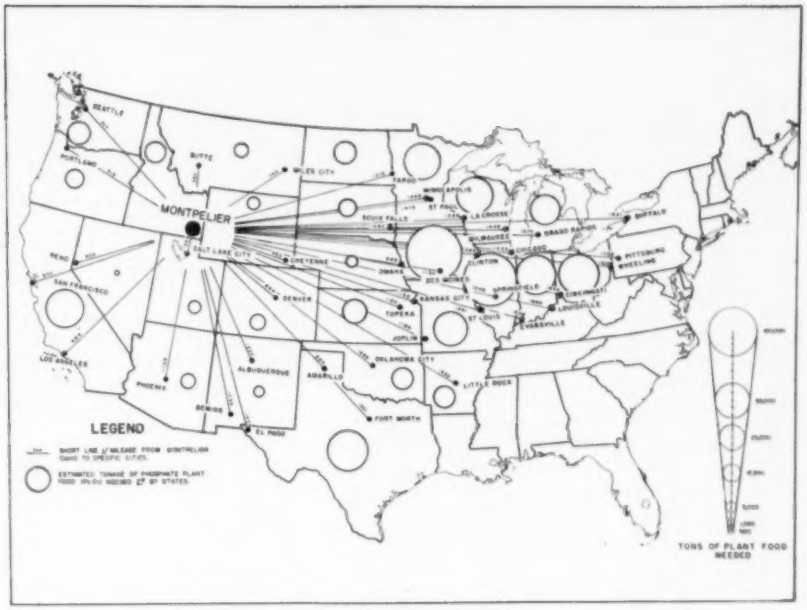

Fig. 2—Rail mileages from Montpelier, Idaho, to important potential markets for western phosphate products.

Source: Legend: [1] Southwestern Traffic Manual, St. Louis S.W. Ry. Lines, Nat'l Ry. Publication Co. [2] State Production Adjustment Committees, Prod. and Marketing Admin., U.S.D.A. 1947. In these cases annual soil phosphate removal by crops was substituted. U.S. Dept. of Interior, Western Phosphate Program, Economic Studies by Bonneville Power Admin., June 1947.

agronomist knows that phosphoric acid is present in the soil in very small quantities, and that it is removed by crops in important but varying amounts, and is frequently a critical plant nutrient element. It is especially critical in the production of the legume crops essential to good crop rotation and soil conserving cropping systems. Phosphoric acid once removed from the soil is replaced primarily in the form of commercial fertilizer. The removal of P_2O_5 by crops can be calculated and can be fairly accurately estimated over broad areas. This, however, does not give an accurate index of the critical shortages in the soil or the areas where fertilizer application will pay. It does give a clew as to the *ultimate need* for fertilizer.

Soil Phosphate Requirements: The agronomists of the West have done considerable experimental work on fertilizer use and know something about the needs of particular soils. The farmers of the West have in recent years, under the stimulation of the agricultural conservation program and the trials established by the colleges and the fertilizer companies, tried out fertilizer in many areas. Estimates of fertilizer needs have been compiled by counties and states throughout the United States and these give some index of future fertilizer needs. These figures are, with all of their limitations, interesting and significant although probably not more than 75 pct accurate.

Fig. 3—Location of phosphate fertilizer plants in United States, 1947.

Soil scientists have made analyses of many soils both by chemical and biological tests. However, these analyses are notoriously misleading in some cases. This is especially true in the wide variety of soils which occur in the western United States.

All of the above indices of fertilizer need point to a continued increase in the fertilizer needs of the western soils.

The estimated "need" for fertilizer may be used as an indication of possible future market requirements, table I. It is apparent that in 1946 consumption of P_2O_5 was only about half of the estimated need in the western area. The estimated need was roughly half of that which is removed annually in crops. It would, therefore, appear that the area is one in which added fertilizer use is very probable if not inevitable in future years. The rapid increase in fertilizer use in this area has been stimulated partly by the high crop price levels. However, it has been associated with a decrease of general production throughout the area where fertilizers have not been applied. In much of the area the high fertility of the virgin soils is nearly exhausted. It is also significant to note that in the older farming areas of the United States, such as the southwestern states, fertilizer application must be double or treble the removal by crops if productivity is to be maintained at high levels.

This rapid rise in the consumption of fertilizer has made it impossible for western manufacturers to produce all of the fertilizer required. Consequently, there has been importation from eastern plants to meet the western needs (fig. 5). Actually

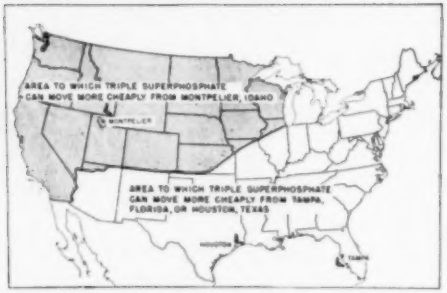
AREA TO WHICH TRIPLE SUPERPHOSPHATE CAN MOVE MORE CHEAPLY FROM MONTPELIER, IDAHO
MONTPELIER
AREA TO WHICH TRIPLE SUPERPHOSPHATE CAN MOVE MORE CHEAPLY FROM TAMPA, FLORIDA, OR HOUSTON, TEXAS
HOUSTON
TAMPA

Fig. 4—Approximate trade areas for western- and southern-produced triple superphosphate.

Freight rate schedules from Montpelier based on those in effect June 30, 1946. U.S. Dept. of Interior, Western Phosphate Fertilizer Program, Economic Studies by Bonneville Power Admin., June 1947.

Table I. Phosphate Fertilizer Consumption, Need, Removal by Crops, and Manufacturing Capacity

	United States	17 Western States	4 North Central States	27 Eastern and Southern States
Fertilizer Consumption (Tons P_2O_5)				
1935-39	703,926	37,338	89,621	576,967
1944	1,288,302	135,172	198,745	954,385
1945	1,338,062	170,106	200,006	967,950
1946	1,534,704	199,682	254,378	1,080,644
1947	1,716,897	239,021	294,916	1,182,960
Fertilizer "Need" (PMA Estimates) (Tons P_2O_5)				
	2,863,296	457,142	880,449	1,525,705
Fertilizer Removal by Harvested Crops (Tons P_2O_5)				
	1,699,064	869,829	316,348	512,887
Fertilizer Manufacturing Capacity (Tons P_2O_5)				
	1,987,000	85,000	200,000 (est.)	1,702,000

some fertilizer originating in Florida, processed at Baltimore, and shipped to mixing plants in Portland, Oregon, finds its way into the markets of Idaho and Montana.

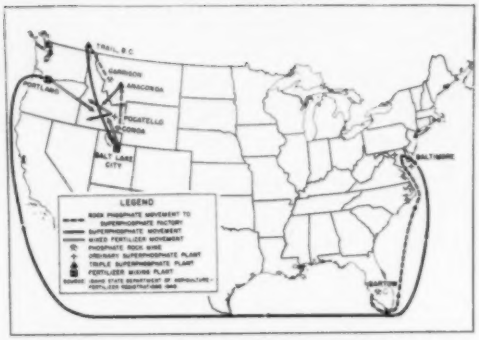

Fig. 5—Movement of fertilizers containing phosphate into Idaho, 1946.

U. S. Dept. of Interior, Western Phosphate Fertilizer Program, Economic Studies by Bonneville Power Admin., June 1947.

The existence of the nation's greatest supply of raw materials, together with the nation's most rapidly growing market in the western United States, has naturally attracted attention as to the possibilities of the future growth of the industry in the West. Apparently two important factors basic to the industry are present in the western area and if the other factors are present or can be provided, production can be economically expanded.

Manufacturing Processes: Phosphate rock may be converted into soluble phosphate fertilizer by treatment with sulphuric acid or by smelting in an electric or blast furnace. The sulphuric acid treatment may result in the production of normal or ordinary superphosphate which contains 18 to 20 pct P_2O_5, or triple superphosphate may be produced which contains 40 to 50 pct P_2O_5. The electric or blast-furnace processes result in the production of phosphorus which may be converted to phosphoric acid and treble superphosphate, ammonium phosphate or other derivatives.

In the western area, within comparatively short distances from the phosphate deposits, there are potential sources of reagents or energy for the production of phosphates and fertilizers by any one of the three methods. Metalliferous sulphide ores

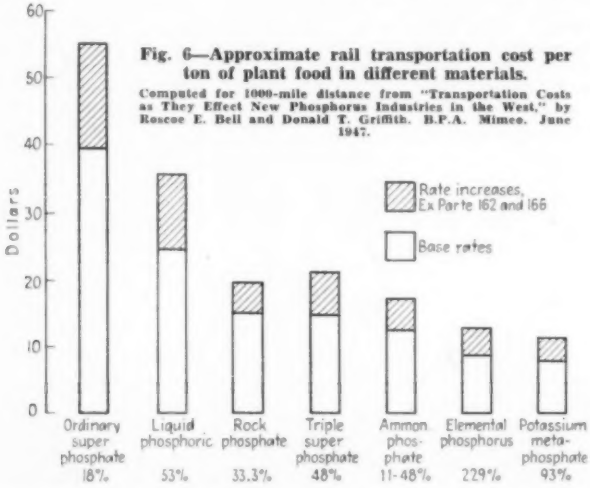

Fig. 6—Approximate rail transportation cost per ton of plant food in different materials.

Computed for 1000-mile distance from "Transportation Costs as They Effect New Phosphorus Industries in the West," by Roscoe E. Bell and Donald T. Griffith. B.P.A. Mimeo. June 1947.

provide byproducts which can be used for the production of sulphuric acid. The Pacific Northwest is the nation's area of greatest potential low cost hydroelectric energy. Coking coal is also present in the intermountain area of Wyoming and Utah relatively close to phosphate deposits. It is, of course, possible to ship rock phosphate from the West to the midwestern United States for conversion to fertilizer. The location of the industry is dependent upon the relative costs of combination of the factors of production and the transportation to market.

It is apparent that the large markets are a rather long distance from the western phosphate deposits. Distances of 900 to 1500 miles from the western phosphate area to important market centers are common. Thus transportation costs become an important factor in reaching the markets economically, and therefore the more concentrated fertilizers have greater transportation advantage. In looking toward future markets it is essential to produce the type of a product which can reach the consumer at a low price (fig. 6). Thus the comparison of the delivered cost of ordinary superphosphate and treble superphosphate shows that in the western and midwestern markets the delivered cost of treble superphosphate is considerably lower than ordinary superphosphate per unit of plant food. It has frequently been pointed out that treble superphosphate is more expensive to produce than ordinary superphosphate. There is often a big difference between manufactured cost and delivered cost, table II. The various items which come in between the manufacturer and the consumer are more related to bulk than to plant food content, thus the lower production cost of ordinary superphosphate is more than offset by the higher delivery costs, as is shown by a comparison of delivered costs of plant food at selected points in the western trade area, table III.

Table II. Effect of Fertilizer Concentration on Retail Cost

Cost Item	Cost per ton of P_2O_5	
	Ordinary[a] Superphosphate 18 pct P_2O_5	Treble[b] Superphosphate 43 pct P_2O_5
Fertilizer f.o.b. factory	$ 82.00	$ 86.50
Bags and bagging at $3.00 per ton	16.70	7.00
Freight at $5.80 per ton	32.00	13.50
Sales at $4.00 per ton	22.00	9.00
Total	$152.70	$116.00

[a] O.P.A. ceiling Pocatello, Idaho MP R 205.
[b] O.P.A. ceiling for treble, minus 7.62 freight allowance apparently included in ceiling delivered to jobber. MP R 205.

Table III. Retail Ceiling Price Per Ton of Phosphate Plant Food (P_2O_5) in Ordinary Superphosphate and Triple Superphosphate, at Various Points, 1945

City, State	Ordinary Superphosphate	Triple Superphate
Portland, Ore.	$175.00	$127.21
Pocatello, Ida.	169.45	120.93
San Francisco, Calif.	154.17	120.93
Fargo, N. D.	166.11	134.30
Minneapolis, Minn.	158.33	131.05
Madison, Wis.	144.45	129.67
Kansas City, Mo.	156.11	124.78
Des Moines, Iowa	158.33	129.55
Chicago, Ill.	138.89	128.00

Source: O.P.A. Maximum Price Regulation 135, 2nd Rev. Nov. 21, 1945.

There is another aspect of the future fertilizer market which is extremely significant. More than half of the fertilizer used in this country goes into mixed fertilizers which are produced by mixing the so-called "simples" such as superphosphate, muriate of potash, and ammonium sulphate. The trend in manufacture of mixed fertilizers is toward higher analysis as a means of delivering the product to the farmer at a lower cost per unit of plant food. High analysis mixed fertilizers cannot be produced with low analysis "simples" such as ordinary superphosphate. The manufacturing capacity of this country available for the production of triple superphosphate is far below the amount of phosphate fertilizer used in mixed fertilizers. Thus the big market for future development is in the production of concentrated fertilizers. This is especially true in the West where shipping distances are great and where an industry pattern of ordinary superphosphate fertilizer plants has not already been established.

Possible Locations of Factories: Granting the need for additional western manufacturing capacity for high analysis fertilizer, what are the most probable alternatives as to type and location of plants? The following deserve some consideration: (1) sulphuric acid process, manufacturing near the phosphate mines, (2) sulphuric acid process manufacturing near smelters where sulphuric acid manufacturing plant capacity can be established, (3) electric furnace processing near the hydroelectric plants on the Columbia River, (4) electric furnace processing near the phosphate mines, and (5) blast furnace processing near the phosphate mines.

The distribution of western phosphate rock deposits, present and potential sulphuric acid plants, and hydroelectric facilities would require shipping of rock or sulphuric acid or transmission of electricity about 300 miles to assemble the factors of production (fig. 7). A comparison may be made of the economy of transportation of sulphuric acid and rock phosphate or transmission of electric energy over these distances. It is apparent that it is more economical to transport rock phosphate to the acid than to transport acid to the rock. It is, also, more economical to transport sulphur than to transport acid and considerably more economical to transmit electricity than to transport either rock or acid. An important advantage of location of the processing plant close to the phosphate mines is that it provides an opportunity to utilize low-grade phosphate rock which otherwise could not be used economically.

It is evident then that the alternatives which deserve the most consideration for western plant establishment are the sulphuric acid process at one of the smelters such as Salt Lake City, Anaconda, or in the Coeur d'Alene district in northern Idaho, an electric furnace plant or a blast furnace plant near the phosphate mines.

In the midwestern market western phosphate would be in competition with Florida phosphate fertilizer produced by the wet process using Texas or Louisiana sulphur as a source of sulphuric acid. Another alternative to consider is the possibilities of manufacturing concentrated fertilizer at a midwestern point using Florida (or western) rock and Texas or Louisiana sulphur. Western phosphate rock cannot reach the midwestern market as cheaply as Florida rock because it is more expensive to mine, it may require more expensive preparatory treat-

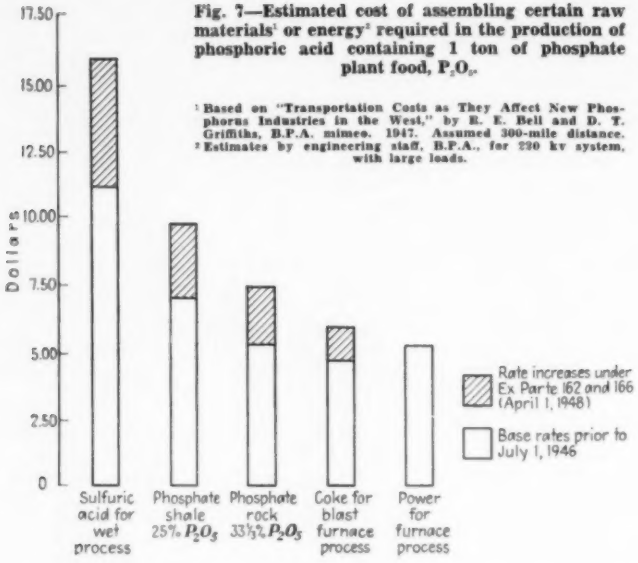

Fig. 7—Estimated cost of assembling certain raw materials[1] or energy[2] required in the production of phosphoric acid containing 1 ton of phosphate plant food, P_2O_5.

[1] Based on "Transportation Costs as They Affect New Phosphorus Industries in the West," by R. E. Bell and D. T. Griffiths, B.P.A. mimeo. 1947. Assumed 300-mile distance.
[2] Estimates by engineering staff, B.P.A., for 230 kv system, with large loads.

ment, and it does not have the advantage of water transportation that Florida rock has.

Cost Comparisons: A comparison of possible costs of manufacturing and transportation indicates that if cheap byproduct acid is available in the western states, the concentrated superphosphate produced at a western point will be able to reach the large midwestern markets at a slightly lower cost than Florida treble superphosphate produced as outlined above. The comparative cost of transportation and production of treble superphosphate at a midwestern point is considerably greater than production in the West or in Florida. Fertilizer produced in the West by the electric-furnace process with power at 2¼ mills (0.225 cents) per kw-hr is estimated to be slightly higher in delivered cost than wet-process production of treble superphosphate in Florida. The production of elemental phosphorus and its transportation to a midwestern point for use in the manufacture of phosphate fertilizer offers a possibility of reducing the costs of levels roughly equivalent to the costs of plant food in Florida-produced treble superphosphate. The difference in the cost of transportation of the various fertilizer products and elemental phosphorus is very great with ordinary superphosphate, the most expensive and elemental phosphorus cheaper even than triple superphosphate.

Conclusion: In the foregoing comments I have attempted to point out:

1. That the potentialities of the western phosphate industry are great, especially if the fertilizer can be produced at a cost that will enable western-produced phosphate to enter the midwestern area.

2. That a western concentrated fertilizer industry can provide lower cost fertilizer to the farmers in the midwestern area than can be provided by local production in that area of ordinary superphosphate from Florida rock and Gulf Coast sulphur.

3. That under average conditions which may be expected to exist in the West, western-produced, concentrated phosphate fertilizers can compete with those produced in the South. This is especially true if additional supplies of low cost byproduct sulphuric acid becomes available in the vicinity of the phosphate deposits.

The production of phosphate by the electric-furnace method has possibilities if low-cost power becomes available in the vicinity of the phosphate deposits. It is important, however, to point out that there are possibilities, because of the wide variation in western phosphates, of setting up a plant under considerably better than average conditions. The most important opportunities for developing better than average conditions are:

1. In the discovery of rock phosphate deposits, which can be mined at low cost through open-pit mining methods.

2. Reducing the cost of rock phosphate at the plant through mining thick beds and beneficiation of the low-grade ores.

3. Reducing the net cost of rock phosphate through byproduct recovery. Vanadium values in western phosphates may run to several dollars per ton of rock.

4. In the use of low-grade phosphate rock in the electric furnace, cost may be reduced by selecting rock which does not require sintering or other pretreatment before furnacing.

Any one of these factors has sufficient influence on the final cost of fertilizer to have a very important bearing on the economics of the operation. There is sufficient variation in conditions in the western phosphate area to make it profitable to search for the most favorable deposits and to develop processes which result in the lowest net cost of production.

These are challenges to you as mining and metallurgical engineers. Upon the answers which you work out to these questions depends the future of the western phosphate industry. The industry has wide horizons and there are great opportunities for development of this western industry in the service of the entire nation through more economical food production and conservation or rebuilding of the nation's soil resources.

Corrosion Resistant Materials and Coatings in Trail Chemical Operations

by E. A. G. Colls

Discusses corrosion in Trail chemical plants producing ammonia, sulphuric, nitric and phosphoric acids, ammonium phosphates, sulphate and nitrate, together with miscellaneous allied material problems and their solution using erosion or corrosion proof materials. Conclusion, that for work of this nature, most costly material may well be cheapest in the long run.

IN all branches of the chemical industry, corrosion plays a very costly part unless it is suitably combatted, and as a result it is probably correct that chemical and design engineers are more corrosion conscious in the field of chemical industry than in others.

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Most metals occur in nature in combination with other elements. The refined or reduced state to which man converts them for his use, by removing the combining elements, leaves them in an unstable condition. Therefore in most of the media to which the metals are exposed they are constantly combining with the constituents of their environment, or in other words they are continually being corroded. Similarly, nonmetallic materials are corroded by undergoing chemical change.

The rate at which corrosion takes place varies widely from metal to metal and medium to medium.

In practically all cases the corrosion of metals in solutions (electrolytes) is electrolytic in nature and the rate of corrosion is dependent on many factors such as temperature, pressure, internal stresses, galvanic action, metal-ion concentration, and so on.

In the chemical industry substances must be handled which are particularly corrosive to materials of construction. At the chemical plants of The Consolidated Mining and Smelting Co. of Canada, Ltd. (otherwise known as Cominco), situated at Trail, B. C., Canada, the following are some of the materials that have to be handled:

Gases. Sulphur dioxide, sulphur trioxide, carbon dioxide, carbon monoxide, oxygen, hydrogen fluoride, ammonia, hydrogen sulphide, and oxides of nitrogen.

Liquids. Solutions of ammonium sulphate, ammonium sulphite, ammonium nitrate, caustic soda and potash, sulphuric acid, nitric acid, hydrofluosilicic acid, hydrofluoric acid, phosphoric acid, aqua ammonia, and formic acid.

Solids. Gypsum (containing phosphoric acid), ammonium sulphate, ammonium nitrate, phosphate rock, caustic soda, potash, and phosphate fertilizers.

It should be noted that most gases are inert when dry and at normal temperatures and pressures but may become extremely corrosive at elevated temperatures. Also, most solids are inert at normal temperatures.

In the preceding section, some of the materials handled in Cominco's chemicals and fertilizers division at Trail have been listed. In order to obtain a clear picture of the corrosion problems encountered there, a brief résumé of the chemical processes involved in fertilizer production will be given.

One of the basic chemicals, sulphuric acid, is produced from SO_2 (formerly waste) derived from lead-smelter and zinc-roaster operations. The acid is obtained either by an indirect process which includes absorption of the SO_2 in ammonium sulphite solution followed by acidification and production of 100 pct SO_2, or by the direct contact acid process. To obtain ammonia, another basic chemical, nitrogen and hydrogen must be produced first. The nitrogen is separated by liquefaction and distillation of air or is produced from nitric-acid plant tail gas (formerly waste) by burning out the oxygen with hydrogen. The hydrogen is produced from distilled and alkalized water by electrolysis or from the oxygen-steam blast coke producer. The mixture of nitrogen and hydrogen under high pressure and high temperature forms synthetic anhydrous ammonia. From the combination of the ammonia with the sulphuric acid is produced crystalline ammonium sulphate, a finished fertilizer.

Another range of fertilizer products is manufactured by first dissolving phosphate rock with sulphuric acid to make phosphoric acid, and then, after separating out the waste gypsum, neutralizing this acid with ammonia to make a series of ammonium phosphate fertilizers.

Finally, in other plants, ammonia is burned with air and oxygen to produce nitric oxides and these by absorption in water, are converted into 56 pct nitric acid. The nitric acid so produced is neutralized under closely controlled conditions to form ammonium nitrate, and a strong solution of this nitrate is sprayed down a tower countercurrent to an uprising current of air to form a useful prilled (pelleted) form of ammonium nitrate that makes an excellent fertilizer product. This product is named "Nitraprills."

The products mentioned above, ammonium sulphate, ammonium phosphates, and ammonium nitrate, are primary fertilizer products. Some secondary products are produced also and as these also present corrosion problems, very brief mention should be made of them. They are aqua ammonia (the solution product of ammonia in water and used at Cominco's for absorbing SO_2), oxygen (the by-product of hydrogen production by the electrolysis of water and of the air liquefaction and distillation process), H_2S (produced in the gasification of coke), and fluorine (as SiF_4 or H_2SiF_6 evolved when mixing sulphuric and phosphoric acids in the phosphate plant, the fluorine being derived from the phosphate rock).

The combatting of corrosion in the production of the foregoing fertilizers or in the production of the simpler compounds entering the various processes, imposes very major considerations of economy, utility, and safety. The attack upon mild steel tanks, piping, valves, fans, and other equipment can be so rapid when handling corrosive liquids or gases that it would be quite uneconomic to operate a process using such corrosion-susceptible materials of construction. In the same way, the usefulness of many materials is limited because of possible pollution or degradation of a product in a vessel or container in which corrosion byproducts are evolved. Furthermore, under conditions of high temperature, high pressure, or high vacuum, the materials of construction naturally play a most important part. If, for example, high-pressure piping or other high-pressure equipment is subjected to internal and unseen corrosion, disastrous results involving extreme hazards to personnel and equipment will sooner or later result. One is, therefore, always faced by such problems in the chemical industry and, while for some purposes no completely satisfactory material of construction is yet available, progress in this direction is constant and increasingly encouraging.

Cominco maintains an elaborate testing and reporting procedure by which existing materials are under constant study. New materials are tested under both laboratory and plant conditions as soon as they appear on the market, and changes to new materials with superior qualities are being made constantly. At the same time, our Research and Development Division is engaged upon the same problems from both fundamental and practical angles. The materials available and of proved utility and economic cost used in Cominco's chemical plants today, cover a wide range of products. These include special paints and cements, brick, sprayed metals, carbon compounds, Vinyon cloths, wood, glass, hard and soft rubber, neoprene, flint, ceramics, transite, quartz, calorized pipe, and the metals lead, aluminum, zinc, silver, copper, nickel, tin, admiralty metal, and the stainless steels. In addition, of course, there are the commoner materials of construction which are still useful for certain applications, mainly cast iron, mild steel, bronze, and so on. Perhaps the simplest way of indicating the practical application of corrosion-resistant materials in the chemical plants is to treat the individual plants or processes in turn.

Sulphur Dioxide Recovery and Acid Plants: In the SO_2 recovery plant and the acid plants at Trail, the commonest form of construction for Glover towers and wash towers handling around 6 to 7 pct SO_2 gas makes use of lead, lined with acid-proof brick. For absorbing SO_2 gas, whether weak (1.5 pct) or strong (6.5 pct SO_2), lead-walled, wood-packed towers are used. Ammonium sulphite solution coolers are constructed of aluminum tubes in steel tanks. More recently "Alrocked" aluminum tubes have been used or the complete cooler has been constructed from aluminum. For storing ammonium sulphite solution, wood-stave, lead-lined tanks are used. For coke-packed driers, steel construction lined with lead and acid-proof brick is employed. For storing ammonium sulphate solution (40 pct) and for pump tanks, wood-stave, lead-lined construction is used. For pumping ammonium sulphate solution under high head, two-stage 275 psi pumps are constructed from A.I.S.I. type 316 stainless steel. Coolers for SO_2 gas are made from standard cast-iron pipe, and acid converters and heat exchangers are constructed with mild-steel shells and trays or tubes. In some cases, calorized pipe is used for the hotter parts of this particular equipment. The 93 pct sulphuric acid is stored in ordinary mild-steel tanks, which have a life of about 18 years. Generally speaking, lead is quite satisfactory for the

cold acid up to 80 pct and for the hot acid up to 70 pct. In the higher acid-concentration ranges, carbon steel and cast iron and high-silicon iron are useful.

A high chromium-nickel steel containing silicon, molybdenum, copper, and a small amount of manganese, looks like a promising material for acid pumps, although no plant operation has yet been experienced with this.

Hydrogen and Ammonia Plants: In the hydrogen and ammonia plants, corrosion problems due to caustic, oxygen, CO_2, formic acid, copper ammonium formate, hydrogen, aqua ammonia, and hydrous ammonia are encountered.

The alternating current to direct current power conversion for the purpose of hydrogen production is accomplished by various types of mercury arc rectifiers. In some of the rectifier cooling water systems not more than one half of one per cent of sodium di-chromate to inhibit corrosion of the steel bowls is used.

For the production of distilled water necessary for making up the electrolyte in the hydrogen plant, extremely pure distilled water is required, and to this end Escher-wyss water-distillation plants are used in which the heat-exchanger tubes are tinned. They have an excellent life. The remainder of the distilled water is recovered from condensed steam and the total distillate fed to the hydrogen cells is required to have a conductivity not higher than 20×10^{-6} mho and as low as possible a content of solids of magnesium and calcium.

The caustic soda or potash used in these cells must be pure to avoid corrosion effects which can be very marked in an electrolytic cell. It is specified that caustic soda contain not more than 0.02 pct chlorine and not more than 0.013 pct sulphate radical. The potash is specified to contain not more than 0.03 pct chlorine and not more than 0.013 pct sulphate. Our patented Trail type hydrogen cells have concrete cell tops fabricated from best grade portland cement with suitable reinforcing, the cement being mixed with crushed quartz. By using concrete collecting bells rather than metal ones, we have thus avoided one of the worst corrosion problems in a hydrogen cell. Our hydrogen-cell gas mains are now made up from transite pipe in lieu of steel pipe which was used formerly. The electrodes in the hydrogen cells consist of mild-steel plate, in the case of the cathodes, but of heavily nickeled mild-steel plate in the case of the anodes. Apart from the undesirably high over-voltage that would result from using uncoated mild steel for both cathode and anode, the corrosion encountered would be entirely uneconomic.

Ceramics also find quite a use in scrubbing towers and wash towers for packing rings and so forth.

In the ammonia plant itself, while low-carbon steel is the standard material for high-pressure piping, valves and other equipment, there are special materials for certain uses. High-pressure forgings for the actual ammonia-synthesis converters are made from nickel-chromium, medium-carbon forging steel corresponding to A.I.S.I. type 3130. In mixed-gas compressor intercoolers, either steel, galvanized iron, or admiralty metal (70 pct copper, 1 pct tin, 29 pct zinc) are used. The internal baskets, heat exchangers, and so on, in the converter employ either mild-steel baskets and tubes or 4 to 6 pct chromium-steel tubes and tube plates.

High-pressure gaskets are of considerable importance because of the need for minimizing leakages. Unless the lens-type steel-ring gasket is used, soft annealed silver or copper or aluminum gaskets prove very useful. Silver in the soft annealed condition is the most useful gasket material for both the large high-pressure forging joints or for smaller joints and valves. Aluminum gaskets, while not seriously affected by anhydrous ammonia, are seriously attacked by aqua ammonia and, in this case, Monel metal finds useful application.

Cooling water may present problems; for example, a curious coincidence was once encountered in the ammonia plant where very severe corrosion conditions were found to be present, apparently due to the cooling water. Corrosion in this case was so severe as to give considerable concern, but it was finally traced down to bacteria in the water, namely Crenothrix. By chlorination of the water, however, the bacteria were killed and corrosion from this cause was stopped.

For building up corroded piston rods, shafts or other pieces of such equipment, the metal spray gun is now proving a useful tool, and mild steel, lead, brass, copper and stainless steel are sprayed. In fact any metal that can be drawn into wire or produced as a powder, can be used for this purpose. Plastics also are now entering the field in this connection.

Phosphate Plant: The plant with the worst and most diverse corrosion problems found in the chemical operations, is undoubtedly the phosphate plant, but even here very satisfactory materials are available. Considerable reliance is placed upon lead and stainless steel. Lead, both hard and soft varieties, is used for launders, agitators, pipelines, vacuum-filter lining, fluorine strippers, and many other pieces of equipment. Both the phosphoric acid reaction agitators and the large Dorrco vacuum filters are lined with 20 lb lead. The former are of wood-stave construction lined with hard sheet lead and Duro acid-proof brick, but a more recent improvement on this type of construction used hard lead walls, brick-lined, and supported by an external skeleton of steel bands. In an even later improvement, but still under construction, are similar agitator tanks of A.I.S.I. type 318 stainless steel. While the life of a hard, lead-lined, wood-stave tank runs from 15 to 16 years, and the steel skeleton lead-lined tanks will probably last longer, the stainless steel tanks, if properly protected against erosion, should last almost indefinitely. The large Dorrco vacuum filters which separate 30 to 33 pct phosphoric acid from gypsum are of steel frame construction lined with soft lead for protection of the shell. On this lead lining, wooden corrugated panels are placed as filter-cloth backing medium to provide channels for the acid to escape through, and on these wooden panels are placed panels of fairly coarse Vinyon cloth overlaid by finer weave Vinyon as the actual filter medium. These panels are held in place by stainless steel A.I.S.I. type 318 straps.

Swenson evaporators on phosphoric acid service also have hard lead walls but the tube bundles therein are built up of Karbate tubes. Karbate is graphite bonded by plastic.

The fertilizer reaction agitators in which phosphoric acid is neutralized by ammonia to produce an ammonium phosphate slurry are built of wood staves, and lined with lead and acid-proof brick, but the shafts and impellers used to promote agitation are constructed of A.I.S.I. type 318 stainless steel and the ammonia inlet pipes are of A.I.S.I. type

316 stainless steel. Further uses for lead in this plant are found not only in the form of sheet material for wrapping conduit, pipe, valves, and so on, which may come in contact with corrosive liquids, but also for phosphoric acid pipelines and for flooring and stair treads in certain wet areas. Stainless steels of the A.I.S.I. type 316 or 318 are used extensively for building impellers, for mechanical agitation, or for the moving parts of Wilfley or La Bour sand pumps.

Another corrosion-resistant material used extensively in the phosphate plant is rubber, both soft and hard. It is used for covering or lining. In grinding the phosphate rock (in 20 pct phosphoric acid), the process is carried out in Allis-Chalmers pebble mills. These mills of steel construction are first lined with rubber. On this lining ¾ in. maple is set in place and then 6x5x10 in. blocks of silex, which are set in acid-proof cement. These blocks are extremely abrasion-resistant and are of a flint-like texture. The actual grinding medium used consists of 4 to 5 in. diam flint pebbles.

In the reaction between sulphuric and phosphoric acids, fluorine as silicon tetrafluoride is produced and this is stripped from the mixed acids by air agitation. The stripper is a hard lead chamber. The effluent gases containing combined fluorine are scrubbed in an unlined, wood-stave tower using 20 to 22 pct H_2SiF_6 scrubbing liquor. The ducts employed for carrying the silicon tetrafluoride gas and the pump tanks involved in the equipment are rubber lined, and the thickeners used consist of wood with rubber lining. Pumps, pump lines, and spray nozzles used in this part of the plant are built of Monel metal or copper. The product, H_2SiF_6, is shipped from the plant in a tank car which is rubber lined.

Sulphate Plant: Turning now to the sulphate plant, in which ammonia and sulphuric acid are reacted together to produce ammonium sulphate in a solution, which is then concentrated by vacuum evaporation, centrifuged, and dried, simpler but nevertheless continued corrosion problems are encountered. The ammonium sulphate solution in the first place is stored in wood-stave, lead-lined tanks and is pumped from there using Wilfley A.I.S.I. type 316 stainless steel pumps into very large Krystal type vacuum evaporators and crystallizers. The evaporator section is built of steel plate on which neoprene lining has been placed. Installation of this lining has to be a very perfectly executed job to insure against blistering under the high (24 in. mercury) vacuum. The crystallizer section of the equipment is also of steel construction but is lined with Goodrich triflex rubber. This material has, incidentally, stood up for some 13 to 15 years with only occasional patching being necessary. Neoprene itself has given excellent service in the evaporators. It is entirely likely that if and when new Krystal units were to be built, they would be constructed throughout of stainless steel. The mother liquor and crystals circulated through the foregoing circuit are pumped by Morris rubber-lined pumps equipped with A.I.S.I. type 316 stainless steel linings. The centrifuge baskets also consist of stainless steel. In this plant, considerable use is made of hard lead for floor covering to collect possible spills or overflows which can be returned to circulation. In addition acid-proof brick set in acid-proof cement has proved a satisfactory flooring. In the sulphate plant, as an experiment, electrical conduit has been installed using aluminum tubing.

Nitrate Plants: In the nitric acid and ammonium nitrate plants, A.I.S.I. type 347 stainless steel is being used almost exclusively for handling 56 pct nitric acid in converters, absorbers, preheaters, boilers, condensers, and storage tanks. In the nitrate plant itself, in the evaporators, preheaters, and neutralizers in which nitric acid is made to react with ammonia to form ammonium nitrate (up to 94 pct strength) type 347 stainless steel is used. This material is also used in the strong liquor tanks and pump tanks. However, the drying is carried out in mild steel, steam heated driers. The prilling (or pelleting) of the concentrated nitrate solution is carried out in 100 ft towers. These towers present a problem in that the use of wood is of course precluded due to the hazards of nitration, and mild steel corrodes so fast that one is left with the option of using either stainless steel as a lining on other materials or of using suitably treated concrete. Aluminum sheathing is also a satisfactory lining. We have used all three methods of construction. For nitrate-plant floors extensive use has been made of acid-proof brick set in acid-proof cement which gives a very pleasing appearance in addition to being extremely durable and easy to keep clean. When making ammonium nitrate for explosive purposes, glass-lined crystallizers are used. These are horizontally-revolving, mild-steel shells lined with glass.

Summary: It is rather interesting to note that out of the large number of stainless steels available today, we are, in the main, dependent on only three types. These are A.I.S.I. type 316 which is an 18/8 steel containing molybdenum, A.I.S.I. type 318 which is similar to type 316 but contains columbium as a stabilizer, and A.I.S.I. type 347 which is an 18/8 steel stabilized with columbium, but containing no molybdenum.

The columbium stabilized steels are necessary where welding has to be carried out on the steel and subsequent heat treating is impracticable, which, incidentally, is generally the case.

Extra low-carbon stainless steels such as A.I.S.I. type 304 E.L.C. and type 316 E.L.C., have recently appeared on the market and may render stabilization with columbium unnecessary. While we do not as yet have any equipment in operation fabricated from these steels, we are investigating their possibilities.

One other unusual material might be mentioned and that is the special wood used for a specific purpose in our recirculated cooling water towers. For this purpose California redwood appears to be ideal. It has stood up in these towers for many years with no apparent ill effects from the continual wetting and drying.

Other materials too numerous to mention having highly specialized application must of necessity be omitted in a paper of this length. However, it is apparent from the foregoing that chemical plants are inherently dependent upon the very best materials of construction most suited for the particular purpose in hand, and that without these materials the life of equipment and the maintenance costs become altogether uneconomic. There can be no question that in general from the chemical plant operators' point of view the best material available for any particular job at once becomes the cheapest material for that job, and this is strikingly borne out by actual maintenance and replacement costs.

Use of an Induced Nuclear Reaction for the Concentration of Beryl

by

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John Dasher,

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A new sorting process for beryllium minerals is described. This depends upon emission of neutrons upon irradiation by gamma rays, a nuclear reaction which is specific for beryllium at the appropriate energy level. Signals given by neutrons when ore passes on a belt are changed to mechanical commands through amplifiers and other electrical equipment.

WHEN beryllium is bombarded by gamma rays under suitable conditions it evolves neutrons. This nuclear reaction,

$$Be^9 + \gamma \rightarrow n + Be^8, \qquad [1]$$

is utilized in the process that is described in this paper. The equation, which physicists often abbreviate to $Be^9(\gamma, n)Be^8$, shows that a photon, γ, is absorbed by the Be^9 isotope of beryllium which then emits a neutron, n, leaving Be^8 as a residue.* It has been determined that the threshold for this reaction is 1.63 mev; that is, gamma rays having an energy less than 1.63 million electron volts do not have enough energy to remove a neutron from the beryllium nucleus. The threshold value for the beryllium (γ, n) reaction is lower than that for any other element. The next most sensitive element is heavy hydrogen (deuterium) whose gamma neutron reaction, D^2 (γ, n) H^1, has a threshold of 2.2 mev. Thus, gamma rays having an intensity between 1.63 and 2.2 mev will evolve neutrons from beryllium and from beryllium only.

Nuclear reactions can be considered as occurring between a projectile and a target. The probability of the occurrence of any such encounter can be expressed in terms of the "cross section" of the target nucleus for the projectile and projectile energy. This cross section, then, is the area normal to the path of the projectile controlled by the target for the purpose of the reaction under consideration. Cross sections are usually expressed in barns, in which a barn is 10^{-24} sq cm. The cross section for Eq 1 is very small, even for nuclear reactions, and of the order of 10^{-3} barns. It is therefore necessary that a high flux of gamma radiation at the appropriate energy be used to obtain an appreciable neutron yield.

In view of the nuclear properties of beryllium it seemed that if pieces of beryllium ore on a moving belt were bombarded with an intense beam of gamma rays, those pieces which are rich in beryllium minerals would emit neutrons at a sufficiently high rate to make it possible to detect that emission, and to use it to actuate a mechanical device for removal of the active from the barren particles.

The principal economic source of beryllium is the mineral, beryl. Beryl occurs mainly in pegmatites which are characteristically coarse grained. The suitability of mechanical concentration depends upon the size at which the desired mineral can be liber-

* This new beryllium isotope Be^8 is unstable and decays spontaneously and rapidly to two helium atoms, but as this reaction is of no importance in the process described in this paper, the matter is mentioned merely for the sake of completeness.

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ated. Fortunately this desired mineral with its specific nuclear reaction often occurs in coarse crystals which would tend to make utilization of the reaction practical.

A series of experiments was conducted to determine if this idea could be reduced to practice.

Detecting Neutrons: Even if the photon used in Eq 1 has no excess energy above the threshold, the neutron emitted leaves the residual nucleus at high speed. Methods for detecting *fast* neutrons are quite inefficient, and it was therefore considered that the neutrons evolved from beryllium should be slowed down, or *moderated*, before detection was attempted.

In order to moderate the neutrons evolved, a hydrogenous material was desired. Plastics and rubber might have been used, but for our experiments a paraffin wax block with a minimum wax thickness of 2 cm between beryl and the detector was shown to work efficiently.

One method of detecting slow neutrons is absorption by some element such as manganese which yields a radioactive product. This method, however, requires integration over an appreciable time and would not be useful for picking.

Another method of detecting slow neutrons involves a fission counter which consists of a foil plated with a thin coating of U-235. Slow neutrons are detected by causing fission which gives a detectable pulse from the foil counter. Because of the other uses for fissionable material it was doubtful if this type of counter could be justified for the purpose of concentrating beryl.

The method adopted for detecting slow neutrons involved the use of counters filled with boron trifluoride gas. The B^{10} isotope, which normally comprises 18 pct of natural boron, undergoes the following reaction:

$$B^{10} \ (n, \alpha) \ Li^{7}. \qquad [2]$$

An energy of 2.3 mev is released in the reaction and the corresponding pulse is easily detected. Moreover, the cross section is so large (3525 barns for B^{10}) that several per cent of the neutrons entering a counter are absorbed by the boron atoms. Various types of BF_3 counters were obtained and tried. Best results were obtained with counters of 1 in. diam and 10 in. long filled to a pressure of 28 to 50 cm of mercury with BF_3 gas enriched in B^{10}.

Since some weak pulses may arise from action of the high gamma flux on the counters, a pulse-discriminating step must be included in the amplifying circuit. The instruments used in transforming the neutron signal to an electrical pulse of standard size and shape included, therefore, a preamplifier, a fast linear amplifier, and a pulse-discrimination-level circuit.* Similar techniques have already been used in radioactivity measurements where alphas are counted in the presence of a high beta activity.

Sources of Gamma Radiation: To obtain the intense gamma radiation desired, the simplest possibility was to use radium. Although radium does not emit gamma rays, its daughter-element products do. About 7 pct of the gamma rays from radium and its descendants have an energy of 2.19 mev which is just under the deuterium threshold, and a total of 27 pct of the radium gammas have energies in excess of 1.63 mev, the beryllium threshold.

Another possible source of gamma rays is radioactive isotopes produced in a pile. Of these, Sb^{124}, with a 60-day half life and 1.72 mev gamma, appeared to be the most promising. Radium was the more economic source when this investigation started. However, multi-curie Sb^{124} sources are now available from pile irradiation of natural antimony at less than $100. A sample of Sb^{124} is now in use.

A third possibility involves the use of Van de Graaff X ray generators. Small Van de Graaff generators which give a two-million-volt X ray (2 mev γ) are in use at a number of hospitals. A larger instrument, adjustable to give up to 4 mev gammas was available for intermittent use at MIT. Initially radium was used for convenience and subsequently the equipment was tested under the Van de Graaff generator.

Results of Experiments with the Mineral Stationary: The gamma rays came from a 1-g radium source (1 curie). The radium source was surrounded by an annular container filled with beryl and quartz, and the radium and beryl were placed in a hole in a wax block which contained the counter tubes. To

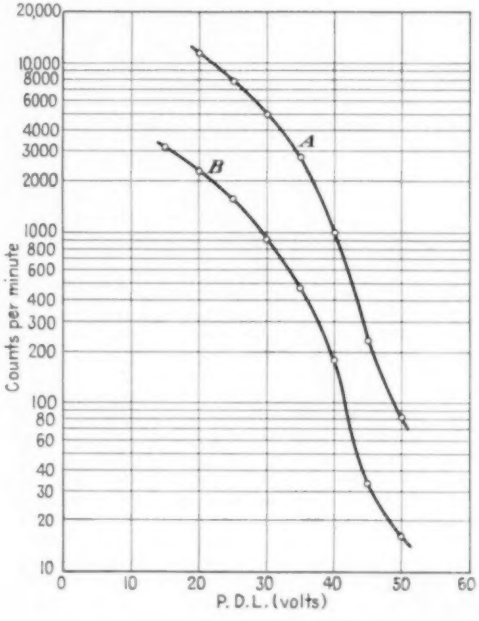

Fig. 1—Bias curves with first 1-g radium source.

A. Neutrons from 500 g pure beryl and background.
B. Background due to radium alone.

* The first experiments utilized a model 501 amplifier, model 500 preamplifier, model 1090 5000-volt negative power supply (Instrument Development Labs., Chicago) and model 101A scaling circuit (Atomic Instrument Co., Boston). For convenience, and in anticipation of better results, the polarity of the H-V supply was changed at a later date. This necessitated changes in the preamplifier and it was decided to make one similar to the Bell-Jordan type AIB (W. H. Jordan and P. R. Bell, *Review of Scientific Instruments* [1947], **18**, 703). A Glassmike condenser was used in the H-V filtering network and a ceramic condenser for coupling the input. A number of preamplifiers of this type have been made and all were entirely satisfactory. When a model 204A (Atomic Instrument Co.) amplifier became available, it was used in place of the 501 as it contained a rise-time selector. Connections from the 204A were as follows: discriminator output to scaler input, high-level output to oscillograph input. Acquisition of a Dumont 248-A oscillograph, containing a triggered sweep of adjustable time, was of considerable aid in all of the neutron-counting work.

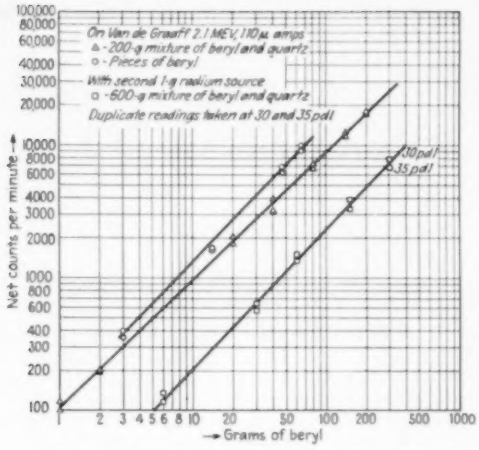
On Van de Graaff 2.1 MEV, 110 μ amps
△ - 200-g mixture of beryl and quartz
○ - Pieces of beryl
With second 1-g radium source
□ - 600-g mixture of beryl and quartz
Duplicate readings taken at 30 and 35 pdl
30 pdl
35 pdl
Net counts per minute
Grams of beryl

Fig. 2—Relation between neutron counts and quantity of beryl.

avoid interference from gammas, various settings of the pulse discrimination level were used in an attempt to obtain the maximum ratio of neutrons from beryllium to background. The result of one such experiment is shown in fig. 1. It is seen that as the pulse discrimination level (pdl) was increased, the ratio of beryl counts to background remained constant; that is, the background was not due to the small pulses caused by the gamma radiation. Upon further investigation this background was found to be due to the fact that the 1-g radium source emitted neutrons, and the high background experienced was due to true neutron counts. Experiments were then made with this apparatus with the MIT Van de Graaff generator. With this source of gammas, a negligible background was obtained.

After the discovery that neutrons were emitted by the first radium source, this source was exchanged for an especially purified source. The second 1-g radium source was found to emit only 10 pct as many neutrons as the first. With the second source, counting rates for pure beryl 50 times as high as the background could be obtained, while with the first radium source, counting rates were only 5 times the background.

Experiments were made with various pieces of beryl and mixtures of beryl and quartz. Results are shown in fig. 2 which shows that the counting rate is directly proportional to the weight of beryl (slope of one on the logarithmic plots). For the Van de Graaff, counting rates of about 100 counts per minute per gram of beryl (approximately 10 pct BeO and 4 pct Be) were obtained. With the second radium source, counting rates were 20 per min per g. In these experiments only one counter was used. Although picking might have been accomplished with the second radium source, it was decided to conduct picking experiments under the Van de Graaff generator.

Picking Experiments: A picking machine was constructed including a wax block, four counters, and preamplifiers connected by coaxial tees to the electronic circuits. The block was mounted on a frame and a semicircular tunnel was provided through it for a 3-in. wide rubber belt. In the middle of the wax block there was a hole for the X ray beam to irradiate the pieces of beryl. On the belt at the point of irradiation was a kicker operated by a solenoid which was actuated by the electronic circuit. The kicker had a maximum rate of 5 cycles per second. The electronic circuit was set in the most sensitive position so that only 1 or 2 neutron counts were needed to actuate the kicker. In this position there were also 3 random kicks per minute caused by spurious counts. The belt was operated at speeds of 6 and 10½ in. per sec which gave irradiation times in front of the 2-in. long kicker of 1/3 to 1/5 sec. Pulses from the counters and preamplifiers were fed to the linear amplifier and the pulse-discrimination-level circuit of the scaler. The scaler input was also sent to an RCA rate meter and relay circuit which actuated the solenoid. This rate meter was far more intricate than necessary for this job as it was set to operate the kicker on one pulse or on two pulses in 1/5 sec. Fig. 3 shows a diagram of the circuit.

Various sized pieces of beryl ranging from 1 to 10 g each were placed on the belt 2 in. apart, and, after the generator was started and adjusted, the belt was started and the pieces passed in front of the kicker. The results obtained are given in table I. Note that by increasing the current and decreasing the distance from the target, 1-g pieces were picked with certainty at the higher belt speed with irradiation times of 1/5 sec per piece.

Next, mixtures of beryl and quartz were prepared. The sizes tested were minus ¾-plus ½-in. and minus ½-plus ¼-in. Various methods of feeding were tried but none was developed which would give a steady feed to the belt. The experiments were conducted with a Jeffrey vibrating feeder fitted with a wooden V-shaped trough in the feeder pan and a stirring device incorporated in the hopper. The feeding rate was adjusted by a variac, but the feeder frequently jammed. Speeding up the vibrator dislocated the jams, but then caused excessively heavy feeding momentarily. When these piles passed in front of the guides used to align the pieces before they reached the kicker, about 10 pct of the material fell off into the discharge chute. This difficulty could probably have been remedied by mechanical im-

Table I. Results of Picking Sized Beryl

Run	Belt Speed, In. per Sec	Target-Beryl Distance, Cm	Current μ amps	Weight of Pieces, G, Each	Number of Pieces		
					Feed	Conc.	Tailing
1	6	90	100	15	3	3	0
2				10	6	6	0
3-6				5	24	23	1
7-8				3	13	10	3
Rerun tailings				3	3	2	1
9-11				1	21	9	12
Rerun tailings (runs 9 and 10)				1	8	6	2
Rerun tailing (run 10 rerun)				1	1	0	1[a]
12-14	10.5	90	100	5	24	22	2
15-16				3	17	11	6
Rerun tailing (run 15)				3	3	2	1[a]
17	10.5	58	100	3	8	8	0
18				1	10	3	7
Rerun tailing				1	7	3	4
Rerun tailing (run 18 rerun)				1	4	3	1[a]
19-20	10.5	86	200	1	19	19	0

[a] Each of these three pieces was spectroscopically examined and found to contain ½ normal BeO content.

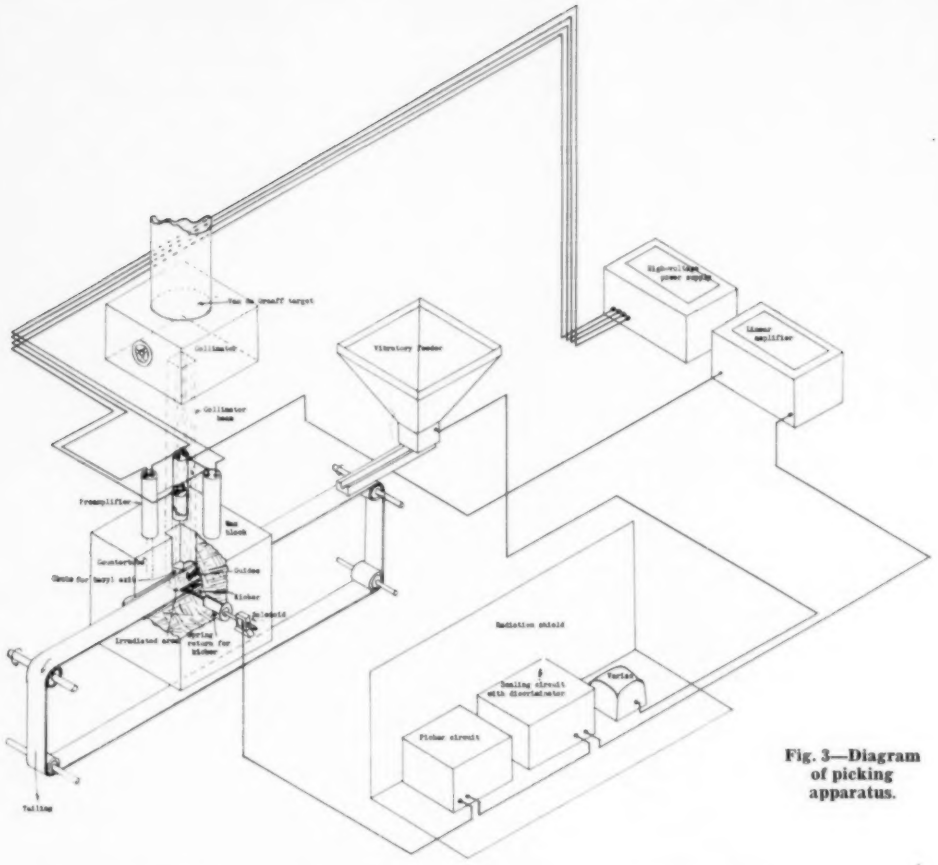

Fig. 3—Diagram of picking apparatus.

Table II. Picking Tests on Mixtures of Minus ¾-Plus ½-In. Pieces

Run	Feed, Pieces		Concentrate, Pieces		Tailing, Pieces		Per Cent Picked	
	Beryl	Quartz	Beryl	Quartz	Beryl	Quartz	Beryl	Total
3	71	300	49	50	23	250	68	27
4	45	450	21	77	24	373	47	20
5	50	859	38	75	11	784	78	12
6	Rerun #5 tailing		9	39	2[a]	745	82	6
7	Rerun part #5 & #6 conc.		30	26	2	64	94	46
8	10	796	9	82	1	704	90	12
9[b]	10	796	9	91	1	705	90	12

[a] Petrographer says these two pieces were feldspar.
[b] Products from 8 recombined and rerun.
Note: Runs 3 and 4: X ray at 2.0 mev and 100 μ amps, counter voltage 1,950, amplifier setting 5,000 gain and 15 pdl, belt speed 6 in. per sec, target to belt 90 cm. Runs 5-9: amplifier at 10,000, generator at 2.1 mev.

provements. The results obtained by picking the various mixtures are shown in table II. Apart from mechanical difficulties of feeding and guiding coarse pieces, the results are promising.

Conclusions: Preliminary experiments have shown that the specific (γ, n) reaction of beryllium can be used for its concentration. Radium, purified from elements such as beryllium so that it emits few neutrons, or a Van de Graaff generator, can be used as a source of gamma rays. Probably a pile-irradiated piece of antimony containing Sb^{124} could also be used. Boron trifluoride counters, preferably filled to a pressure of 28 to 50 cm of mercury with the gas enriched with B^{10}, can detect efficiently the neutrons evolved provided paraffin or other hydrocarbons are used as moderators. Pulses from the counters can be used to operate a mechanical picker.

With the apparatus employed, pieces of beryl as small as 1 g can be picked with certainty at rates approaching 5 per sec. Before the method can be applied practically, a suitable economic source of intense gamma radiation must be developed as well as a mechanical feeding system which will feed coarse pieces of rock at an even rate.

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Concerning the Adsorption of Dodecylamine on Quartz

Using an adsorption-column technique the partition of dodecylamine between quartz and water has been determined at concentrations ranging from 0.5 to 4000 mg per liter. The adsorption varies as the square root of the concentration. Attachment of dodecylamine to quartz is reversible. The amount adsorbed in a flotation operation giving almost complete recovery is under 5 pct of the amount required for a monolayer.

by

A. M. Gaudin

and

F. W. Bloecher, Jr.

THIS paper describes the results of a series of tests carried out to determine the partition of dodecylamine between distilled water and the surface of quartz at various concentrations of dodecylamine acetate in solution. An adsorption-column technique was utilized for this purpose. Ralston and Harwood, of Armour and Co., had synthesized marked dodecylamine acetate for us.[1] In their preparation which we employed, 0.0007 pct of the total carbon was the radioactive isotope, carbon fourteen. Unmarked dodecylamine acetate of high purity was also prepared by them for our use. The presence of carbon fourteen in the dodecylamine enabled us to analyze quantitatively for dodecylamine on the surface of quartz, using a radioactive-tracer technique. The method used was that known as internal gas counting of carbon dioxide, formed by oxidation of the amine. This method has already been described,[2, 3, 4] and the reader is referred to these papers for details of the analytical method and procedures.

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Adsorption Apparatus: To determine the partition of dodecylamine* between the surface of quartz and surrounding liquor, a technique analogous to chromatographic columns was utilized. The experimental setup is shown in fig. 1.

* In this paper the term "dodecylamine" is used to refer to the amine-chain part of the salt and would include the undissociated salt, dodecylamine, dodecylamine hydroxide, or dodecylammonium ion. We are inclined to favor the view that the dodecylammonium ion is the active ingredient, but we do not feel that our data warrant ruling out all other forms of "dodecylamine".

The dodecylamine-acetate solution was made up in a volumetric flask and poured into the glass funnel shown. It was allowed to drip slowly onto the quartz bed, the rate of flow from the funnel reservoir being controlled by a hose clamp. The quartz column was contained in a small, fritted-glass filter (size 20 mm, C). The beaded top edge of the filter was cut off to enable the holder and contents to fit into the open end of the 30/45 joint on the combustion tube. This combustion tube is used in the analytical train described in a previous paper.[5] A piece of gum-rubber tubing joins the bottom of the fritted-glass filter to a filter flask, which in turn is connected to a vacuum line, or water aspirator. The rate of flow of amine solution through the bed was

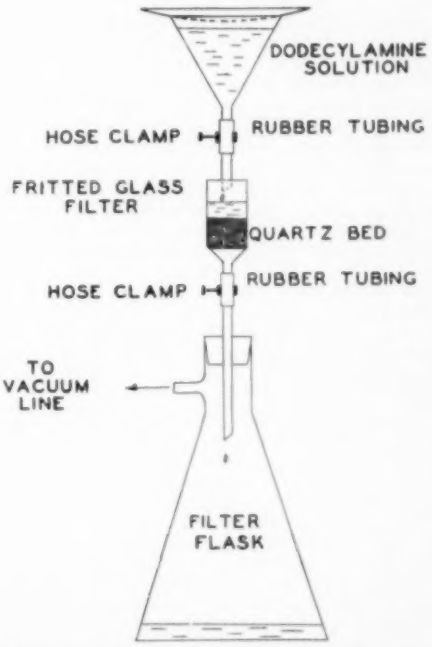

Fig. 1—Schematic sketch of adsorption apparatus.

Table I. Test Description

Test No.	Conc. Amine Acetate, Mg per Liter	Wt. SiO_2, Grams	pH Solution	pH Effluent	Wt Soln. Retained, G	Total Amount Soln., Cc	Exposure Time	
							Hr	Min
32a	0.53	5.3920	6.10	6.14	0.9670	500	4	25
32b	0.53	5.8811	6.10	6.52	0.9757	500	4	40
34a	1.54	6.3340	6.27	Not	1.2240	500	4	20
34b	1.54	6.8312	6.27	recorded	1.1111	500	3	50
35	3.05	6.4685	6.24	Not recorded	1.2457	500	4	30
30a	5.47	7.3651	5.90	6.31	0.9636	500	2	57
30b	5.47	5.6787	5.90	6.27	1.1153	500	3	05
5	11.06	6.9602	5.92	6.17	1.0476	500	2	10
14	32.5	4.3857	6.08	5.92	0.7685	500	2	10
4	62.4	7.9380	5.83	6.22	1.0637	1,000	3	10
2	63.86	7.2017	Not recorded		1.6535	380	1	40
1	64.56	8.1644	Not recorded		0.8536	150	Not recorded	
6	86.0	8.9168	6.28	6.34	1.1917	500	1	30
7	87.0	6.4116	5.93	6.19	0.8747	500	1	40
3a	120.2	5.5492	Not recorded		0.7061	280	2	45
3b	120.2	4.9140	6.2	6.3	0.4922	200	3	00
10a	202.2	5.3339	6.33	6.27	0.6591	490	2	35
10b	202.2	5.1513	6.33	6.38	0.6710	490	3	05
12	443.5	6.0045	6.53	6.45	0.8040	800	3	05
15	632.3	5.4669	6.50	6.48	0.8906	500	2	35
13	662.3	6.4898	6.58	6.44	0.7595	500	2	45
16	993.4	4.2028	6.70	6.70	0.9062	500	4	30
19a	1,394.6	4.2906	6.99	6.88	0.6942	500	3	30
19b	1,394.6	4.3605	6.99	6.68	0.6845	500	4	10
17	1,856.9	4.4849	6.99	6.40	0.5538	500	4	20
18a	3,018.6	4.2837	6.87	6.60	0.5056	500	4	00
18b	3,018.6	4.8977	6.87	6.62	0.6382	500	4	25
22a	3,998.8	0.9988	7.04	6.94	0.2127	250	4	25
22b	3,998.8	1.0245	7.04	7.10	0.3007	250	4	35

controlled by a second hose clamp. The flow was adjusted so that the rate of feed to the bed was equal to the rate of efflux from the bottom, a small pool of liquor covering the quartz bed at all times.

Preparation of Quartz: In order to prevent channeling through the quartz bed, it was felt that quartz of fairly uniform size should be used. Therefore, cleaned quartz was fractionated in an infrasizer, and the cone 5 fraction set aside for use.

Massive quartz was first crushed through a jaw and roll crusher, and a 20/65 mesh fraction then separated by screening. Leaching with mixed hydrochloric and nitric acids cleaned the quartz of abraded iron from the crushing operation. Rinsing with an excess of distilled water freed the quartz of dissolved iron and excess acidity. The acid-washed 20/65-mesh fraction was then ground in an Abbe mill with porcelain pebbles and distilled water. The milled product was dried and then fractionated in an infrasizer.

The cone 5 fraction had an average particle size of 21.5 microns (measured as the perpendicular to the long dimension on a microscope stage). The surface area of the quartz was measured by the B.E.T. method of nitrogen adsorption,* and by the newer method in which krypton gas is used.[7] Since the quartz was relatively coarse, reproducibility of the determinations using the nitrogen adsorption method was poor, two separate runs giving values of 2430 and 2250 cm^2 per g. However, the krypton method is much better suited for particles in this size range, and in three measurements resulted in an average value of 2320 cm^2 per g. This figure has been assumed as the correct specific surface of the quartz.*

* The measurements made with krypton were experimentally reproducible to within about ±2 pct. However, the surface area of the quartz, calculated from the experimental data, depends on the area occupied by each krypton molecule. Assuming a close-packed liquid layer, this area is 14.0 sq Angstroms per molecule. Beebe[8] found, however, that on a "calibrated" sample of anatase, the area was 19.5 sq Angstroms per molecule, indicating that the arrangement of the molecules in the adsorbed monolayer was not independent of the underlying crystal structure. Beebe's value depended upon the prior determination of the anatase surface both by the gas adsorption method of Brunauer and Emmett[9], and the absolute method of Harkins and Jura (*Jnl.* Amer. Chem. Soc. (1944), **66**, 1366.) In the absence of any experimental data giving the surface area occupied by krypton on quartz, we have arbitrarily used an area of 16.8 sq Angstroms per molecule, halfway between the liquid value, 14.0, and Beebe's figure of 19.5.

Test Procedure: In carrying out an adsorption test, an excess of dodecylamine-acetate solution was passed through the miniature column. The quartz surface became covered with adsorbed dodecylamine to an extent determined by the experimental conditions. Among the important variables affecting this partition are temperature, concentration of dodecylamine acetate in the solution, the pH, and the presence of foreign ions in the solution. An attempt was made to keep all these factors constant, except that the concentration of the dodecylamine acetate in solution was varied from test to test. The temperature ranged from 20° to 25°C. No attempt was made to adjust the pH, and for almost a ten thousand-fold change in amine concentration, the pH changed only about one unit, climbing from the distilled water value of about 5.9 up to about 7.0 for very high concentrations of dodecylamine acetate (around 4 g per liter). The quartz was stored in a covered glass jar, and all glassware kept scrupulously clean by scouring with hot, chromic-acid

Table II. Radioassay Results of Tests Outlined in Table I[a]

Test No.	Conc. Amine Acetate, Mg per Liter	Percentage Strength Amine Acetate[c]	Activity Amine Acetate, Cpm per Mg[b]	Activity of Solution, Cpm per Cc	Actual Activity, Cpm (total)[e]	Activity Less B and Soln., Cpm[d]	Total Wt Amine Acetate Adsorbed, Mg	Mg Amine Acetate per Gram SiO_2
32a	0.53	100.0	29,300	15.5	826	767	0.02618	0.00485
32b	0.53	100.0	29,300	15.5	1,115	1,056	0.03604	0.00613
34a	1.54	100.0	29,300	45.1	1,785	1,686	0.05754	0.00908
34b	1.54	100.0	29,300	45.1	1,855	1,700	0.06007	0.00879
35	3.05	100.0	29,300	89.4	2,297	2,142	0.07311	0.0113
30a	5.47	100.0	29,300	160.3	3,905	3,708	0.1266	0.0172
30b	5.47	100.0	29,300	160.3	2,973	2,781	0.09389	0.0165
5	11.66	100.0	31,500	367.3	5,758	5,327	0.1691	0.0343
14	32.5	47.50	14,960	486.2	2,906	2,482	0.1659	0.0378
4	62.4	10.07	3,172	197.9	1,667	1,410	0.4445	0.0558
2	63.86	10.07	3,172	202.6	1,596	1,215	0.3830	0.0532
1	64.56	10.07	3,172	204.8	1,608	1,400	0.4373	0.0536
6	86.0	10.07	3,172	272.8	2,162	1,791	0.5646	0.0633
7	87.0	10.07	3,172	276.0	2,007	1,716	0.5410	0.0844
3a	120.2	10.07	3,172	381.3	1,839	1,524	0.4805	0.0866
3b	120.2	10.07	3,172	381.3	1,429	1,185	0.3767	0.0767
10a	202.2	10.07	3,172	641.4	2,439	1,971	0.6214	0.1165
10b	202.2	10.07	3,172	641.4	2,412	1,937	0.6107	0.1185
12	443.5	3.83	1,200	534.9	1,930	1,458	1.206	0.201
15	632.3	3.708	1,135	717.7	2,196	1,467	1.293	0.250
13	662.3	2.655	939.3	635.3	2,185	1,649	1.719	0.265
16	903.4	2.655	777.9	772.8	1,945	1,152	1.481	0.352
19a	1,394.8	2.023	592.7	826.7	2,295	1,695	2.843	0.663
19b	1,394.8	2.023	592.7	826.7	2,293	1,691	2.853	0.654
17	1,856.9	2.023	592.7	1,100.6	2,293	2,207	3.025	0.853
18a	3,018.6	2.023	592.7	1,789.0	6,825	5,885	9.929	2.32
18b	3,018.6	2.023	592.7	1,789.0	7,665	6,487	19.94	2.23
22a	3,998.8	0.9546	279.7	1,118.0	1,318	1,037	3.708	3.71
22b	3,998.8	0.9546	279.7	1,118.0	1,190	811	2.900	2.83

[a] Assay for test 15 made in counter 7. For tests 16 through 32 counter 4 was used. For tests 13 and 14 counter 10 was used. All other assays were made in counter 8.
[b] Weight per cent active salt in diluted salt. [c] Cpm = counts per minute.
[d] Corrected for resolving time. [e] B = background activity; Soln. = activity from solution retained in voids.

cleaning fluid and rinsing thoroughly with distilled water.

In detail, the test procedure was as follows:

1. An accurately weighed amount of dodecylamine acetate was dissolved in distilled water in a volumetric flask. The tests were usually run in duplicate, one liter of solution being made up. 500 cc of solution was then passed through each column.
2. Four to ten grams of quartz were weighed out in a tarred beaker, dispersed in distilled water, and transferred to a tarred, fritted-glass filter, using as much distilled water as was required to complete the transfer. Suction was applied to the fritted-glass filter during the transfer.
3. The amine solution was passed through the quartz column, using the setup shown in fig. 1. The level of the liquor in the filter was always kept above the top of the quartz bed. A record was kept of the exposure time of quartz to amine solution.
4. About 3 cc of the stock amine solution was taken out for a pH measurement in a Beckman, Model G, pH meter. The meter was standardized with buffer solution before each determination.
5. Near the end of the run, a few cubic centimeters of effluent were collected for another pH measurement, the meter being standardized again before this determination.
6. At the end of the test, the quartz bed was not sucked dry by the vacuum. Instead, as soon as air started to pass through the bed, the filter and contents were removed and weighed. The difference between the final weight and the total dry weight of quartz and filter, was equivalent to the volume of solution (in cubic centimeters) retained in the voids of the column. The radioassay results were corrected for the activity due to this solution retained in the bed.
7. The filter and contents were placed in a combustion tube for oxidation.[6] Before combustion, however, the filter and contents were allowed to stand for a day or so at room temperature in order to dry. Attaching the bottom of the fritted-glass filter to a hose from an air line, and inserting the filter, open end down, into a combustion tube, made it possible for one to gently blow the quartz out of the filter into the bulb. The filter was then pulled up out of the tube, and reinserted with the bottom end down. With the bottom end of the filter down, very little gas collected underneath it during combustion of the sample.

Test Results: Several dozen experiments were made, and of these 29 tests have been retained as being significant. The test data are too bulky to be presented in a single tabulation, and are, therefore, presented in two tables. In table I are pertinent test data exclusive of radioactivity measurements, arranged not in chronological order, but in order of increasing concentration of dodecylamine salt in the feed liquor. In table II are the radioassay results of the experiments, arranged in the same order as in table I.

The headings of table I are self-explanatory. It may, however, be mentioned that col. 7, headed, "Total Amount Solution, Cc," refers to the total volume of solution passed through the bed in the exposure time listed in col. 8.

In table II, the third column headed, "Percentage Strength," refers to the weight per cent of radioactive dodecylamine acetate present in the diluted salt. The activity of the dodecylamine acetate was cut down to a suitable level for each test by proper dilution with the unmarked salt. By dissolving an accurately weighed amount of unmarked dodecylamine acetate in benzene, together with an accurately weighed amount of radioactive salt, complete mixing of the unmarked and radioactive salts was obtained. By cooling the solution to about 10°C the dodecylamine acetate recrystallized and was separated from the benzene by filtration. The percentage strength was calculated by dividing the weight of active amine acetate by the total weight of marked and unmarked salts.

The activity of the dodecylamine acetate used in each test was determined by multiplying the actual specific activity* of the undiluted salt (counts per minute per milligram in a given counter) by the percentage strength divided by 100. For example, the 10.07 pct salt used in test 1 had an actual activity of

10.07 x 31,500 x 0.01 = 3,172 counts per min per mg, where 31,500 is the actual specific activity (cpm per mg) of the undiluted radioactive salt in counter No. 8.

The activity of the solution was calculated by multiplying the specific activity of the salt by the concentration (milligrams per liter), and dividing by 1000. For example, in test 30, the solution activity was 5.47 x 29,300 x 0.001 = 160.3 cpm per cc

Multiplication of the activity per cubic centimeter of solution by the volume of solution retained in the voids of the column gave the activity attributable to the liquid retained in the damp quartz at the end of the test.

The figures reported in col. 6, headed "Actual Activity, Cpm (Total)," are the activities observed by radioassay of the carbon dioxide produced by oxidation of the filter and its mineral contents. These actual activities, in counts per minute, are corrected for resolving time of the circuit.[5] Col. 7 records the net, actual activity; that is, the total actual activity less background and less the activity due to the solution retained in the bed. The total weight of amine adsorbed (reported as milligrams of dodecylamine acetate) was determined by dividing the net actual activity by the activity per milligram of salt. Dividing the total weight of amine salt adsorbed by the weight of quartz, gave the adsorption per gram of quartz.

Graphical Presentation of Data: Fig. 2 is a plot on logarithmic scale of the dodecylamine acetate concentration in solution versus the weight of dodecylamine acetate adsorbed per gram of quartz. Fig. 3 is a plot of the same data expressed as mols of dodecylamine acetate per liter, and mols of dodecylamine acetate per gram of quartz. The curves of fig. 2 and 3 may be termed adsorption isotherms of dodecylamine on quartz, since the temperature at which the adsorption occurred was held practically constant.

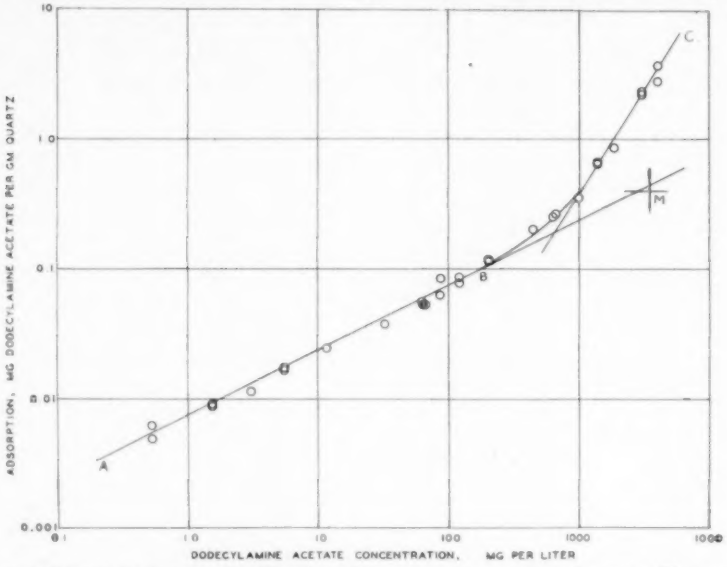

Fig. 2—Adsorption isotherm of dodecylamine on quartz; pH 6 to 7, temperature 20° to 25°C.

These figures show that the adsorption of dodecylamine increases with the solution concentration, but that the adsorption is not directly proportional to the concentration. Instead, the adsorption increases as the square root of the concentration (slope of line *AB* equals ½), up to a solution concentration of about 200 mg per liter. Adsorption of dodecylamine then increases more rapidly, eventually more rapidly than the solution concentration. Such a change in behavior would seem to indicate that at least two mechanisms are involved in the adsorption process we have investigated.

To interpret these unexpected results, we have found it useful to consider, on the one hand, the adsorption that would correspond to the formation of a monomolecular layer of dodecylamine at the quartz surface, and, on the other hand, the solution concentration that corresponds to the formation of micelles.

According to the crystallographic orientation of an element of surface, the shape of the area that can be allotted to each cation-adsorption site is a

* By actual activity is meant the actual number of decays of C-14 atoms occurring during a given time in the sensitive volume of the counter tube. This actual activity is not observed directly by radioassay. It is determined by taking the observed counting rate, correcting that for resolving time of the instruments (period when the scaling circuit is dead and will not record a pulse from an ionizing event in the counter tube), and subtracting from that corrected rate the background activity. See ref. 2 and 5 for complete discussion.

lozenge or rectangle, of such proportions that it does not differ very radically from being a square and has an average area of 23.4 sq Angstroms.[6] It would seem, therefore, as a first approximation, that a monolayer of adsorbed dodecylamine chains would be oriented on the quartz surface in a ratio of one chain to each cation-adsorption site, providing only that the space required for each amine chain were less than 23.4 sq Angstroms. It happens that the cross-sectional area per chain, for close packing of long-chained hydrocarbons on the surface of water, is equal to 20.5 sq Angstroms,[9] or about 15 pct less than the available surface per adsorption site on the quartz surface. We shall therefore assume that a complete monomolecular coating of adsorbed dodecylamine on quartz requires 23.4 sq Angstroms of surface per adsorbed chain.

Since the surface area of the quartz employed in the experiments was known to be 2320 cm^2 per g, the amount of reagent required to form a monolayer can be calculated. Dividing the total area per gram of quartz, in square Angstroms, by the product of the area required per site, 23.4 sq Angstroms, times the number of molecules per gram mol (Avogadro's number), gives the amount of reagent required to form a monolayer, expressed in gram mols per gram of quartz.

$$2320 \times 10^{16} / 23.4 \times 6.02 \times 10^{23} = 1.65 \times 10^{-6} \text{ mols per g quartz}$$

This is equivalent to 0.40 mg dodecylamine acetate per gram of quartz. In fig. 2 this quantity of dodecylamine acetate is represented by the horizontal line passing through point *M*.

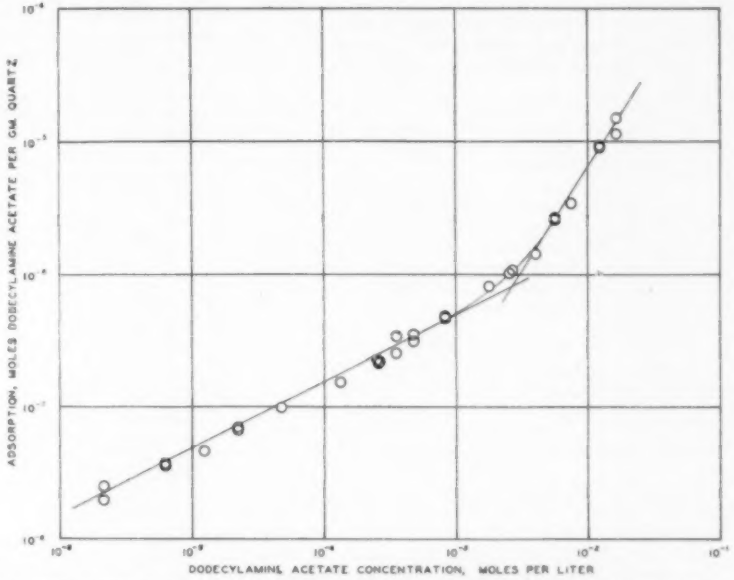

Fig. 3—Adsorption isotherm of dodecylamine on quartz, molar concentrations.

The studies of the late A. W. Ralston and his associates have established that in water solutions of normal, long-chained amines, the dissolved substances exist at low concentrations in a dispersed state, but that an aggregated or micellar condition appears after a certain critical concentration is reached. In the case of n-dodecylamine acetate in water at 30°C, this critical concentration is about 3.5 g per liter.[10] We will accept that as the critical point for micelle formation in the solutions we used. In fig. 2 the vertical line through point *M* is at this concentration. Point *M*, therefore, would represent the coexistence of a critical micellar concentration of dodecylamine salt, and of a completed monolayer of adsorbed dodecylamine. It is reasonable to assume that these two events should happen simultaneously, since both indicate inability of the single phase, represented by a homogeneous solution, to contain more than a certain maximum amount of solute.

It is interesting to observe that point *M* falls close to an extension of the straight-line portion of the adsorption isotherm. This leads to the speculation that the spread between the lines *ABM*, and *ABC*, represents the occurrence of polymolecular adsorption before completion of the initial or monomolecular coating. It is certain that with concentrations of dodecylamine acetate in excess of 1 g per liter, the coating is several chains in thickness. It is postulated here, as a result of the consideration of point *M*, that the coating is not uniformly thick, and that at solution concentrations below that represented by point *B*, is in places entirely lacking. It may furthermore be indicated that at concentrations of dodecylamine acetate below 100 mg per liter, the coating is entirely monomolecular, and is less than one-fifth complete. For example, at a solution concentration of 1 mg per liter, the coating is about 1.8 pct complete.

Attainment of Adsorption Equilibrium: As a rough check to determine how rapidly equilibrium conditions were attained in the adsorption column, tests 1, 2*a*, 2*b*, and 4 were run. The solution concentration in each was about the same, around 63 mg per liter, as was the weight of quartz. The data obtained are summarized in table III.

Table III. Equilibrium Adjustment Data

Test No.	Conc. Amine Acetate, Mg per Liter	Total Amount Soln., Cc	Exposure Time, Hr	Exposure Time, Min	Mg Amine Acetate per G SiO_2
1	64.56	150	Not recorded		0.0536
2a	63.86	380	1	40	0.0533
2b	63.86	100	0	30	0.0325
4	62.4	1,600	3	10	0.0558

Running only 100 cc of solution through the column in 30 min resulted in adsorption of only 0.0325 mg dodecylamine acetate per gram of quartz. Passing 150 cc through the column (time unrecorded) resulted in almost the same adsorption figure as when one liter of solution was passed through the column in 3 hr, 10 min. It was therefore felt that by

using 500 cc of solution for each test, and an exposure time of at least 2 hr, equilibrium conditions in the miniature adsorption column would be attained.

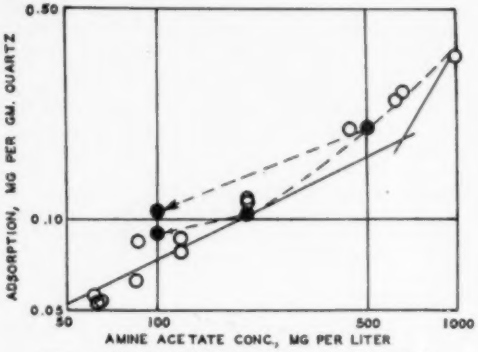

Fig. 4—Graphical illustration of attempt to approach equilibrium from above.

As an additional check, however, an attempt was made to approach equilibrium from above. In one test, the quartz was first coated with dodecylamine at a solution concentration of 506 mg per liter. One liter of 100 mg per liter solution was then passed through the column in 3 hr, 20 min. The adsorption expected from the curve of fig. 2 was 0.074 mg per g of quartz. The actual adsorption was 0.106 mg per g. Evidently the volume of the second solution, and the exposure time of the quartz to solution, were not sufficient to cause complete readjustment of equilibrium.

A second test was run, in which the quartz was first coated with dodecylamine from a solution of 200 mg per liter strength. Then one liter of 100 mg per liter solution was passed through the column in 4 hr, 10 min. Adsorption expected was again 0.074 mg per g, and the actual observed figure was 0.090 mg per g.

A graphical representation of these tests is shown in fig. 4, where a portion of the curve of fig. 2 is reproduced together with the points obtained by approaching equilibrium from above (black circles). It is obvious that dodecylamine goes onto the surface of quartz when equilibrium is approached from below, and comes off when equilibrium is approached from above, but that in neither instance has equilibrium actually been attained. It had been hoped that the exposure time and amount of solution allotted for each adsorption test would be sufficient to insure attainment of equilibrium. The above tests, however, indicate that adsorption equilibrium was approached, but was not quite attained.

Reversibility of Adsorption: If the attachment of dodecylamine to the surface of quartz is an adsorption phenomenon, it should be possible to replace adsorbed radioactive dodecylamine with unmarked dodecylamine, or vice versa, at any given solution concentration. Three tests were run to see if this were possible.

In one test the quartz was first coated with unmarked dodecylamine at a solution concentration of 198.8 mg per liter. Then, 500 cc of radioactive dodecylamine-acetate solution (198.6 mg per liter strength) was run through the column in 2 hr, 35 min. The adsorption expected from fig. 2, on the basis of equilibrated exchange, was about 0.116 mg per g of quartz, and the actual observed adsorption, determined by radioassay, was 0.110 mg per g.

In another test, an attempt was made to replace adsorbed radioactive dodecylamine by unmarked dodecylamine. The surface was first loaded up with radioactive dodecylamine at a solution concentration of 200.2 mg per liter. Then 500 cc of unmarked dodecylamine-acetate solution (200.4 mg per liter strength) was run through the column in 3 hr, 15 min. If complete replacement of the radioactive amine had occurred, there should have been no observed activity (aside from the background count) upon radioassay of the sample. Of a total of 0.118 mg of dodecylamine expected to be adsorbed on all of the quartz, only 0.0042 mg, or 3.6 pct consisted of the original radioactive coating.

A third test was run at a dodecylamine-acetate concentration that is in the range of concentrations encountered in actual flotation practice. The sample was first coated with unmarked amine at a concentration of 4.50 mg per liter, and then one liter of radioactive dodecylamine-acetate solution of the same strength was passed through the column in 3 hr, 25 min. The adsorption expected was 0.0160 mg per g, and the observed adsorption was 0.0152 mg per g of quartz. Assuming the difference to be made up of unreplaced, unmarked dodecylamine, 5 pct of the original, unmarked dodecylamine coating remained on the surface.

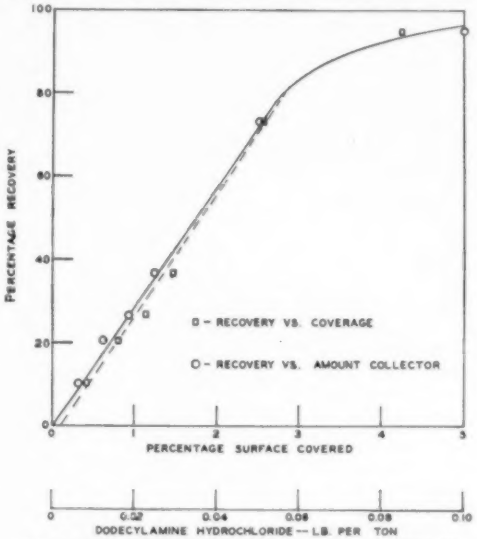

Fig. 5—Flotation recovery of quartz vs. amount of collector added, and vs. percentage available surface occupied by dodecylamine.
Amount of frother constant at 0.08 lb per ton.

Thus, it seems that the attachment of dodecylamine chains to the surface of quartz is a reversible phenomenon similar to other reversible adsorption phenomena. At any one instant, for a given concentration of dodecylamine acetate in solution, there is a certain total amount of dodecylamine on the quartz surface. The individual dodecylamine chains, however, are not firmly anchored there. They are continually arriving at and departing from the surface. At equilibrium, the rate of adsorption from solution is equal to the rate of desorption, or departure of dodecylamine chains from the surface.

Correlation of Adsorption and Flotation Data: In connection with another study, a number of batch flotation tests had been run on the same type of quartz as was used in the adsorption tests. Because of the availability of data on the partition of dodecylamine between quartz surface and liquor, and because of the ability to measure the amount of surface on the quartz by the gas-adsorption method, it now becomes possible to correlate flotation recovery with the density of the adsorbed collector coating. The astonishing fact emerges that to effect even nearly complete recovery the coating is very far from being a complete monolayer.

Of course knowledge of the reagent addition in the flotation tests is expressed directly in terms of pounds of reagent per ton of mineral or in terms of concentration of *added* reagent per unit volume of liquor. The quantity involved in the partition data is not the quantity of reagent added but the quantity of reagent *remaining* in solution at the end of the equilibrating process. It is possible to go from data expressed in one way to data expressed in the other form by a mathematical process of successive approximations which are detailed in the thesis by one of the authors.[11]

The results obtained in correlating density of adsorption with flotation recovery are summarized in fig. 5. For example, a recovery of 95 pct is obtained with the surface covered to the extent only of 4 to 5 pct, and a recovery of about 40 pct is obtained with the surface coverage of only about 1.5 pct.

In making the calculations that led to these results, allowance was, of course, made for the fact that the specific surface of the quartz used in the flotation tests was not the same as that of the quartz used in the partition data presented earlier in this paper. In fact it was nearly twice as large, being 4430 instead of 2320 cm^2 per g.

We have assumed that the equilibrium conditions as represented by our adsorption isotherm were also attained in the flotation cell, and that the surface concentration in terms of amount of reagent adsorbed per unit area of quartz at complete surface coverage was proportional to the concentration depicted by the horizontal line through point *M* in fig. 2.

It is, of course, also implicitly assumed that the reagent ions are standing upright on the surface of the mineral, much as trees in a forest. Clearly, if the reagent ions are assumed to lie flat or at a considerable angle to the vertical, their surface coverage is much greater. Also, if the reagent ions are assumed to be in constant motion, the amount of surface over which they sweep during their vibrations and gyrations about the point of attachment is larger still.

Summary of Results: Using radioassay technique and radioactive dodecylamine acetate, an isotherm for the adsorption of dodecylamine from solution on to the surface of quartz has been established for a pH range of 6 to 7, and a temperature of 20° to 25°C. At low concentrations the amount adsorbed per gram of quartz varies directly as the square root of the solution concentration. The solution concentration that would be required to form a monolayer of adsorbed dodecylamine appears to be equal to the concentration at which micelles appear. At submicellar concentrations, however, a polymolecular adsorption coating begins to form before the monolayer has been completed.

The reversibility of the dodecylamine adsorption process was demonstrated by showing that equilibrium conditions on the surface could be approached from above as well as from below. This reversibility was also demonstrated by replacing adsorbed, unmarked dodecylamine with radioactive dodecylamine, and vice versa.

It was shown that pure quartz, with a specific surface of 4430 cm^2 per g, could be floated almost completely (95 pct recovered), at a pH near 7, with about 5 pct of the total available adsorption sites occupied by dodecylamine chains. This was achieved with 0.1 lb dodecylamine hydrochloride, and 0.08 lb terpineol per ton of quartz. At that reagent figure, about 60 pct of the collector was on the mineral surface, and 40 pct in the liquor. A curve was presented showing the relationship between flotation recovery and percentage surface covered by collector, together with the relationship of recovery to amount of collector added.

Acknowledgments

Armour and Company, through their generosity in setting up the Armour Flotation Research Project, have made this research work possible. H. M. Corley, Manager of the Chemical Division, and Dr. Victor Conquest, Director of Research, of Armour and Company, have been most generous with their help and encouragement. Acknowledgment is due the late A. W. Ralston, and his successor, H. J. Harwood, of the Armour research staff, for their work in preparing the reagents used.

W. W. Miller, of Brookhaven National Laboratories, contributed valuable suggestions concerning the use of the carbon-dioxide-filled counters. The authors would also like to express their appreciation of the suggestions and discussions profferred by Professor P. L. de Bruyn of the Department of Metallurgy at M. I. T.

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Conditioning and Treatment of Sulphide Flotation Concentrates Preparatory for the Separation of Molybdenite at the Miami Copper Company

by C. H. Curtis

THE valuable mineral content of the current feed to the Miami concentrator is as follows: copper, 0.7 pct total; molybdenum, 0.01. Flotation of this ore yields a sulphide concentrate containing: chalcocite, 44 pct; molybdenite, 0.5; pyrite, 50.0; insol, 5.5.

A combination of potassium ethyl xanthate and pentasol amyl xanthate as collectors, and pine oil as frother, are used in this flotation. Rejection of pyrite is encouraged by holding the amount of collectors used to the minimum consistent with copper recovery and by operating at high alkalinity (equivalent to 0.35-0.40 lb CaO per ton solution of pH 11.0).

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AIME San Francisco Meeting, February 1949.

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The molybdenum recovery in the sulphide concentrates under the above flotation conditions is approximately 50 pct of that originally present in the ore. Taking into account the acid soluble molybdenum, indicated molybdenite recovery is 75 to 80 pct.

The attempt to separate the molybdenite into an acceptable molybdenum product begins with the bulk sulphide flotation concentrate just described. This concentrate is composed of chalcocite, whose floatability has been promoted to the fullest extent possible for the sake of its recovery from the ore, together with the pyrite which has been activated along with the copper mineral. The problem is to deaden the copper and iron minerals, and to float the molybdenite. Obviously in the accomplishment of this end, conditioning and preparation of the pulp, prior to flotation, plays an all important role.

The first step is thickening to 50 to 60 pct solids, with milk of lime added to the thickener feed to maintain an alkalinity of the pulp equivalent to a pH of 8.5 to 8.8 during its residence in the thickener. The purpose of the thickening is primarily to reduce the volume of pulp for subsequent treatment. However, the relatively prolonged retention of the pulp in the thickener at the desired alkalinity is known to have a favorable depressing effect upon pyrite. There is a limit for this alkalinity above which a depressing effect upon molybdenite occurs.

The thickened pulp (alkalinity: 0.015 lb CaO per ton, pH 8.8), discharges into an agitator, retention time approximately 2 hr, to which additional lime is added to raise the alkalinity to 0.35 to 0.40 lb CaO per ton solution, pH 11.6. This additional lime is required for pyrite depression and can be tolerated without loss of molybdenite because of the limited time of contact in the conditioner tank.

The pulp leaving the lime conditioner passes through two successive steaming tanks, which are mechanically agitated, and into which live steam is admitted directly into the pulp near the bottom of the tanks. The temperature of the pulp is maintained as near boiling as possible. The steaming time is approximately 4 hr. The pulp leaving the last steamer has an alkalinity of about 0.04 lb CaO per ton solution, pH 8.7. It is believed that oxidation of the copper and iron sulphides occurs during steaming, the resulting sulphates reacting the calcium hydroxide to calcium sulphate and thus reducing the alkalinity. Since the steamer-feed solution is already saturated with calcium sulphate, the calcium sulphate produced during steaming is precipitated. It is believed that this calcium sulphate is precipitated preferentially on copper and iron mineral surfaces thus decreasing their floatability.

Aside from the "lime chemistry" during steaming, pine oil is displaced from the pulp and xanthate decomposed, which has a major effect upon the deadening of the copper and iron sulphides.

Following steaming, the hot pulp is admitted to another conditioning tank wherein it is aerated, primarily for cooling, but incidentally for additional oxidation of the copper and iron sulphides.

The resulting "deadened" pulp is then diluted to 20 pct solids, a specific collector for molybdenite, ordinary stove oil, is added and the separation of the molybdenite by flotation is undertaken at a pH of 8.5 to 8.8 in standard Miami air-flotation machines. B-22 frother is used when necessary. A regrind of the thickened rougher concentrates is made prior to the first cleaning operation chiefly for rejection of insoluble in subsequent flotation. The cleaner concentrate is then stepped up to 90 pct MoS_2 in an 8-cell Denver flotation machine No. 18. Sodium silicate is added to the cleaner circuit. Its effect is to flocculate molybdenite and stabilize the froth.

In summary, it may be stated:

1. Separation of molybdenite into an acceptable product from sulphide copper concentrates by flotation involves preliminary preparation and conditioning of the pulp, which is of major importance.

2. This preparation and conditioning consists of several successive steps: (*A*) Thickening to 50 to 60 pct solids at controlled alkalinity to reduce volume of pulp and to contribute to depression of pyrite. (*B*) Agitation at high-pulp density for limited time with additional lime to provide for depression of pyrite. (*C*) Steaming at high-pulp density for decomposition of xanthate and xanthate surface films, evolution of pine oil, and oxidation of sulphide minerals other than molybdenite. The latter involves sulphating of lime with probable precipitation of calcium sulphate preferentially on copper and iron minerals. (*D*) Aeration at high-pulp density for cooling, and for further oxidation of copper and iron sulphide minerals. (*E*) Dilution of pulp to 20 pct solids and addition of specific collector for molybdenite, common stove oil.

It is hardly necessary to point out that this rather drastic procedure for depression of previously activated copper and iron sulphide minerals, without at the same time depressing molybdenite, is possible due to the inherently high floatability and refractory nature of molybdenite. However, molybdenite is susceptible to depression by excessive lime which must therefore be limited to the amount consistent with satisfactory molybdenite recovery.

The steaming procedure is being carried on at Miami Copper Co. under license agreement with Janney, Nokes, and Johnson, holders of letters patent on the process.

Coal Preparation for Synthetic Liquid Fuels

by W. L. Crentz,
J. D. Doherty,
and E. E. Donath

The principal washing problems in preparing coal for the synthetic fuel industry may be stated as: (1) ash removal for the Bergius process; and (2) sulphur removal for the Fischer-Tropsch process. A study is made of the effect of ash on oil losses during the elimination of the nonliquefiable portions of the coal in the hydrogenation process. The oil loss increases with ash content and results in increased consumption of coal, hydrogen, and power.

IN 1948, the United States used nearly six million barrels of petroleum products every day. Although substitution of synthetic fuels for the natural petroleum product is not here yet, large quantities of special-purpose coal will be required in ever-increasing volume as the domestic production of oil from other sources fails to meet demand and the production of synthetic fuels begins.

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Petroleum economists do not agree as to when the shortages of domestic oil will appear; but the increasing per capita consumption of oil, coupled with a constantly growing population drawing on a potential domestic oil supply estimated to be 85 billion barrels, sooner or later will result in demand exceeding domestic supply. When this situation comes about, the excess demand will be either eliminated by curtailment of consumption or the demand met by increasing importation of foreign oil or by the production of synthetic fuels. The first recourse would definitely retard the heretofore unbroken rise in American living standards, while dependence upon a foreign oil supply during a war or other emergency would be dangerous. On the other hand, the enormous coal reserves in the United States, plus sizable deposits of oil shale, indicate adequate means of augmenting a diminishing domestic oil supply.

Some idea of the impact of an expanding synthetic liquid fuels program on the coal-mining industry may be gathered when it is considered that to supply just half of our present-day petroleum needs synthetically from coal would increase the coal output by about 500 million tons annually, using an 11,900 Btu bituminous coal as the plant feed. If either a subbituminous coal or lignite were used, the quantity required would be materially increased.

Synthetic Fuel Processes: The principal synthetic fuel processes using coal as a raw material are the Bergius, or coal-hydrogenation process, and the Fischer-Tropsch, or gas-synthesis process. These two processes not only produce different end products, but the preparation of the coal used in each presents its own individual problem to the coal-preparation engineer. The hydrogenation process brings together a mixture of dry, finely powdered coal, catalyst, and heavy oil to form a paste. Addition of hydrogen under rather high pressure and temperature conditions brings about transformation of the coal to a liquid product, from which a high-grade gasoline and other motor fuels may be obtained. The Fischer-Tropsch process requires the production of synthesis gas, a mixture of carbon monoxide and hydrogen. At the Bureau of Mines Louisiana, Mo., demonstration plant, this gas is being made by reacting dried pulverized coal with carefully controlled volumes of oxygen and superheated steam. After purification and in the presence of the proper catalyst and suitable conditions of pressure and temperature, the hydrogen and carbon monoxide components of the synthesis gas react to form motor fuel. The two processes are not competitive but rather com-

plement each other to form a well-rounded economy of oil self-sufficiency.

Coal-preparation Problems: The question then arises: What coal-preparation problems will become paramount in order to discharge a satisfactory product from the washing plant to the synthetic fuel-manufacturing plant? While problems of drying and pulverizing the coal fall within the coal-preparation field, this paper will include only the coal-cleaning phase.

At the risk of over-simplification, the washing problems in preparing coal for the synthetic fuel industry may be stated as: (1) ash removal for the Bergius process; and (2) sulphur removal for the Fischer-Tropsch process. The problem of both ash and sulphur reduction is not only technologic but economic.

It will be the task of the coal-preparation engineer to devise the type of coal-cleaning equipment that will enable him to exercise the control necessary to transform float-and-sink data into operating results. A start has been made in this direction by the construction of washing plants designed to prepare metallurgical coal by low-gravity separation. Unlike the preparation of coal for general industrial use where the degree of preparation is determined mainly by competitive conditions, the preparation of synthetic-fuel coal will have a definite monetary value for each per cent of ash reduction in the coal for hydrogenation and each per cent of sulphur reduction for gas-synthesis coal.

In meeting these demands, the coal industry will be faced with the problem of satisfying a new type of customer. It is basic that the qualifying standards as to what constitutes a satisfactory product are set by the consumer and met by the producer. Only during times of a so-called "sellers' market" will the consumer relax his demands for the type of coal best suited to his needs. It is characteristic of the coal industry, with its large productive capacity and numerous operators each struggling for a share of the market, that the demands of the consumer are paramount. Heretofore the yardstick of industrial consumer acceptability has been based chiefly on calorific value. With the possible exception of buyers for metallurgical use, the large customers of the coal producers have been interested in the product primarily from a steam-raising standpoint. Purchasers of coal for conversion to synthetics will be interested not only in the calorific value but to an even greater degree in the adaptability of the coal to processing. Their questions will involve rate of conversion, and ash and sulphur content (not as a heat-value factor but as an item of expense in chemical processing of the coal). The coal industry must recognize the importance of obtaining this new information about the coal resources, so that, when the quality specifications are stated, the preparation-plant operator will be in a position to meet the requirements.

Preparation Practice at the Hydrogenation Plant, Billingham, England: The United States Government in 1943 dispatched a scientific mission to Great Britain to examine the hydrogenation plant of the Imperial Chemical Industries, of Billingham, England. Reports given the American investigators indicated no great difficulty in the preparation of a suitable clean coal, using a sand-flotation process. Raw coal containing 8 to 9 pct ash was crushed to $\frac{3}{8}$ in. top size, dedusted at about 1/16 in., and the $\frac{3}{8}$x1/16 in. product fed to a 10 ft diam sand cone. The float product from this cone is dewatered and desanded over a shaker screen and then fed to a second cone, which makes the final separation at low gravity. Reports vary as to the separation gravity, but float-and-sink data of the raw coal indicate an 82 pct recovery of float coal containing 2.5 pct ash on a specific gravity of 1.35. Using American washing practice as a criterion, the Billingham separation would appear to be formidable; and so it would if the washing characteristics of the British coal were similar to our middle western coals containing the same raw coal ash content. Fortunately for the British, this particular raw coal is adaptable to low-gravity separation much more readily than the medium ash content American coal, the term "low-gravity" being used to indicate separation at a specific gravity substantially lower than is employed in common commercial washing operations. More than 63 pct of their raw coal floated on a 1.30 sp gr which moved the large coal mass at least 0.05 sp gr unit away from the separation gravity, materially cutting down the amount of coal found within $\pm$0.05 unit of the 1.35 washing gravity. Many of the American coal beds will produce large amounts of float coal on 1.35 sp gr, but raw coals having an 8 to 9 pct ash content usually produce a float product containing 4 to 5 pct ash in the 1.35 sp gr range, which frequently defies further deashing.

Effect of Ash in Coal Hydrogenation: In the Bergius process, the effects of ash in the coal are felt in the following manner:

1. Effect upon the catalyst.

2. Some ash constituents tend to settle in the converters, adversely affecting the space time yield and service factor. Periodic converter shut-down to remove ash accumulation might be necessary in some cases.

3. Decrease in oil yield.

Problem 1, the effect of ash on the catalyst, usually can be corrected by added catalysts or chemical treatment. Problem 2 may have great importance but would vary, depending on the properties of the individual coal. Problem 3, decrease in oil yield because of high ash coal, will be examined and discussed more fully later in the paper. Quantitatively, it has been estimated that the oil loss due to ash amounts to about 9 lb of oil for each per cent of ash in 1 ton of the hydrogenated coal. The residue obtained after recovery of oil from the ash may have some value as a fuel, but it is not expected that the fuel value would materially affect the economic balance of loss of oil against the cost of cleaning the coal. An additional benefit derived from reduction of ash by mechanical cleaning is that washing usually decreases the fusain content of the coal, thereby improving the coal conversion.

Fortunately for the preparation engineer, the coal requirements of the hydrogenation process are divided about equally between coal actually hydrogenated in the process and coal used for secondary purposes like steam raising, production of producer gas, and gasification coal for hydrogen. This affords an outlet for a middling product, rich in bony coal, resulting from preparation of a low-ash coal by low specific-gravity separation. The ash specifications of the coal to be used in the Bergius process can best be set at the lowest ash percentage obtainable with a coal recovery of at least 50 pct of the raw-coal feed.

Washability studies of many of the eastern and midwestern coal beds indicate that they contain varying amounts of coal low enough in ash content to meet the ash requirements of the synthetic fuel industry. This low-ash coal has not been separated out in the past, as the small recoverable yield made the process economically unfeasible. Furthermore, separation of the low-ash coal particles from coal particles of slightly higher ash content required plant control outside the range of conventional equipment.

Effect of Ash and Sulphur in Gas Synthesis: In the Fischer-Tropsch (gas-synthesis) process, the quantity of ash in the raw-coal feed is not as critical as in the Bergius process. This can readily be understood because the Fischer-Tropsch process calls for preliminary gasification of the coal leaving the inert high-ash material behind as residue. Nevertheless, the ash matter in the raw coal is not gasified, so that it may be economically advantageous to wash the coal before gasification to increase the volume of synthesis gas per ton of coal charge. Here again, the problem is more economic than technologic, for even though the raw feed is first subjected to mechanical cleaning, either for ash removal or to assure a uniform plant feed, the operation probably would involve well-known washing methods now being used in commercial preparation plants.

Unlike the ash, sulphur in the coal is converted into gaseous products and must be scrubbed from the synthesis gas. Because this is an expensive operation, the bulk of the removable sulphur must be washed from the coal before gasification. Estimates by the Bureau of Mines indicate that the cost of removing sulphur from synthesis gas is $..15 per ton of gasified coal when the sulphur content of the charge is 3 pct. Increase in the sulphur content of the gasified coal from 3 to 5 pct would impose an estimated additional cost item of 37 cents per ton of coal for the removal of sulphur from the synthesis gas. Total cost of gas purification, using a 5 pct sulphur-content coal, amounts to $1.52 per ton of coal. Even the use of low-sulphur coal containing 1 pct sulphur would require a capital investment of almost four million dollars in purification equipment in a gas-synthesis plant producing 10,000 bbl per day and a total of 75 cents per ton of coal gasified, or 34 cents per barrel of oil for the removal of the sulphurous products. Of course, the recovered sulphur may have a market value which can be applied against the cost of removal, but even under the most favorable pricing system the cost of gas purification will in most instances justify cleaning the coal. Thus it is easy to see the economic advantages of cleaning the coal used in the Fischer-Tropsch process to a low-sulphur content if the sulphur can be removed by mechanical cleaning. Examination of coals in Ohio, Indiana, Illinois, and western Kentucky shows that the sulphur content of those available in large quantities is high, in some instances exceeding 5 pct in the raw coal. Even after intensive treatment for sulphur reduction, the sulphur content of the washed coal would be in the range of 1.5 to 3.0 pct. Within these normally high-sulphur areas, there are rather sizable deposits of low-sulphur coal. The low-sulphur and low-ash bed condition of the Illinois No. 6 coal in the Franklin-Williamson district is well-known. Other areas in the region undoubtedly are responsive to sulphur reduction at low-gravity separations.

Sulphur reduction for gas-synthesis coal appears to be a much more difficult problem than ash reduction for the hydrogenation process. Many of the high-sulphur coal beds contain significant amounts of sulphur in those forms that are not removable by gravity separation. Very fine crushing will reduce the sulphur content so that the float product will contain little more than the organic sulphur. Where the amount of organic sulphur in the coal remains excessively high, the problem of further reduction by chemical means is outside the coal-preparation field. The financial return to the conversion-plant operator through the decrease in cost of expensive gas-purification equipment seems large enough to encourage the coal-preparation industry in developing methods to reduce the sulphur content of the clean coal to a minimum. In the low-gravity range, separations of many American coals have shown that significant sulphur reduction does not occur until the separating gravity is lowered to about 1.30. With these coals, the sulphur content in the 1.35 sp gr float product is approximately as great as would result from a 1.60 sp gr separation. Even at the low separation point of 1.30 sp gr, the yield of float material would be in the range of 50 pct of the raw feed. Unlike the Bergius process, the Fischer-Tropsch does not require sizable quantities of auxiliary fuel. This poses a problem in the disposal of a middling product. About 90 pct of the coal requirements for the gas-synthesis plant is used in producing synthesis gas, therefore the small yield of float coal resulting from the low-gravity separation would have to be justified almost entirely on the basis of the decreased sulphur content, as the plant requirements for a secondary fuel are not sufficient for utilizing the large amount of middling material produced in the washing process.

Uniformity of Plant Feed: Perhaps one of the most rigid requirements will be uniformity of the process fuel for both the hydrogenation and gas-synthesis methods. The addition of oxygen and steam to produce synthesis gas undoubtedly will be a closely controlled operation that will necessitate a fuel uniformity seldom imposed by the average fuel user. Likewise, in the hydrogenation process, variations in ash content and petrographic constituents would materially affect and change plant operating conditions. To meet this requirement, it may be necessary to wash certain coals that appear satisfactory in other respects.

Economics of Low-ash Coal in Hydrogenation: In coal hydrogenation, an important operating factor due to the ash content of the coal is loss of oil in eliminating the ash and other nonliquefiable portions of the coal after the first stage of the process. The loss increases with ash content of the coal and results in increased consumption of coal, hydrogen, and power.

To determine the economic limits for cleaning coal for hydrogenation due to this one factor, the Coal-to-Oil Demonstration Branch of the Office of Synthetic Liquid Fuels, Bureau of Mines, studied the increased oil yield with decreasing ash content of the coal. A comparison was made of the economic advantages resulting from the use of 2.3 pct ash coal from Rock Springs, Wyo., and a 6.0 pct ash coal from the same mining field in a 30,000 bbl per day hydrogenation plant. It was discovered that using the lower-ash coal produced a savings of 2.8 pct of coal equivalent to hourly savings of 6.5 tons of coal on the moisture and ash-free basis. More important

than this saving in coal consumption is the reduction in capital costs for the 30,000 bbl per day plant by 3 million dollars when using the lower-ash coal. This saving of 3 million dollars is further reflected in the daily plant operating costs and amounts to $1400. per day or $58. per hour. With this information, it is then possible to calculate the total savings when the ash content of the hydrogenation coal is reduced from 6.0 pct to 2.3 pct ash and from those calculations to estimate the highest permissible washing cost in cents per ton for each unit ash reduction that would be justified by the decreased coal consumption and the reduction in plant operating costs. The value assigned to the 6.5 tons of moisture- and ash-free coal saved, using the 2.3 pct ash coal instead of the 6.0 pct ash coal, would depend on the cost of coal to the plant on an as-received basis. Assuming the value of the ash- and moisture-free coal to be $2.00, $4.00, and $6.00 per ton (this being equivalent to 7, 14, and 21 cents per million Btu) the 6.5 tons of coal per hour saved would result in an hourly dollar savings of $13., $26., and $39., respectively. Adding these figures to the fixed savings in plant operating costs of $58. per hour, the total hourly savings would become $71., $84., and $97., depending on the value assigned to the coal. Dividing these savings by the 230 tons of moisture- and ash-free coal processed hourly in a 30,000 bbl per day plant reveals a savings of 31, 37, and 42 cents per ton of coal. These reductions in cost occurred through reduction of ash content from 6.0 to 2.3 pct, or 3.7 pct. This figures to a maximum allowable cleaning cost of 8.4, 10, and 11.4 cents per ton of process fuel for each 1 pct reduction in ash content. However, these figures are based on moisture- and ash-free coal, so allowances must be made for the ash and moisture in the coal as purchased and for the washery loss in cleaning the coal.

Table I shows the maximum allowable cleaning costs in cents per ton of coal as received for each per cent of ash reduction, using coals containing 10, 20, and 40 pct ash plus moisture and a washery recovery of 50 and 70 pct of the raw feed as clean coal. A large proportion of the remaining 50 and 30 pct, respectively, may be recovered as middlings and utilized as powerhouse coal.

These calculations consider the decreased fuel value of this middling product when used in the powerhouse. This adjustment is reflected in the aforementioned hourly savings of 6.5 tons of moisture- and ash-free coal when operating with the 2.3 pct ash content Wyoming coal.

Table I. Maximum Allowable Washing Costs for Different Coals for Each Unit of Ash Reduction, Cents per Ton of Raw Coal, As-received Basis

Yield of Washed Coal, Pct	50			70		
Ash Plus Moisture in Raw Coal As-received, Pct	10	20	40	10	20	40
Maximum allowable cost of washing for each unit of ash reduction, cents per ton:						
Based on moisture- and ash-free coal valued at $2.00 per ton (7 cents per million Btu)	3.8	3.4	2.5	5.3	4.7	3.5
Based on moisture- and ash-free coal valued at $4.00 per ton (14 cents per million Btu)	4.5	4.0	3.0	6.3	5.6	4.2
Based on moisture- and ash-free coal valued at $6.00 per ton (21 cents per million Btu)	5.1	4.6	3.4	7.5	6.6	5.0

Using a cost figure of 15 cents per ton of coal feed to the washery as the operating cost of the washing plant, it is possible to calculate the minimum reduction in ash content for different coals to justify washing the coal. These data appear in table II.

Table II. Minimum Reduction in Ash Content to Justify a Washing Cost of 15 Cents per Ton of Raw Coal, Per Cent

Yield of Washed Coal, Pct	50			70		
Ash Plus Moisture in Raw Coal As-received, Pct	10	20	40	10	20	40
Reduction in ash content necessary to justify a washery cost of 15 cents per ton of feed coal:						
Based on moisture- and ash-free coal valued at $2.00 per ton (7 cents per million Btu)	4.0	4.4	6.0	2.8	3.2	4.3
Based on moisture- and ash-free coal valued at $4.00 per ton (14 cents per million Btu)	3.4	3.8	5.0	2.4	2.7	3.6
Based on moisture- and ash-free coal valued at $6.00 per ton (21 cents per million Btu)	2.9	3.2	4.3	2.0	2.3	3.0

These data, showing the beneficial effect of decreased ash content in the hydrogenation coal, reflect the advantages in only one phase of the process. Washing the coal usually decreases its fusain content and might materially improve the coal-to-oil conversion period. In addition, ash constituents tend to settle in the converters, with harmful effect. The influence of the ash content of the coal with reference to these latter two factors will depend on the properties of the individual coal and can only be determined by experimentation. Undoubtedly, when these benefits have been evaluated, the economy of the coal cleaning will be enhanced.

Board Discusses Variety of Subjects at February Meeting

Seven hours were spent on AIME business by the Board of Directors at the Annual Meeting, and many important matters were reviewed. Twenty of the 26 elected Directors were present at the Sunday meeting, which began at 10 a.m. and wound up shortly after 5 p.m., with an hour out for lunch. Section delegates and Directors elect were also on hand, as were seven of the ten Divisional Chairmen, who are Directors ex officio. Dr. Young presided. Monday afternoon the incoming Board of Directors met in executive session immediately following the annual business meeting of the Institute, with President McLaughlin in the chair.

A few pertinent figures from the Treasurer's report for 1949, with comparable data in parenthesis for 1948, follow: Income from dues, $225,973 ($208,015); initiation fees, $16,888 ($14,562); advertising, $85,721 ($68,344); publication sales, $20,889 ($28,727); miscellaneous income, $4,069 ($9,064); total income, $393,133 ($328,711). Expenses for publications, $300,364 ($236,916); Institute expenses (rent, library, Local Sections, Divisions, Branches, meetings, Secretary's traveling, insurance, pension) $83,137 ($104,871); General Office expense (salaries not otherwise allocated, accounting, membership, telephone, stationery, postage, furniture, miscellaneous $90,450 ($80,157); total expense, $473,951 ($424,862). The income figures do not include $8,769 ($11,493) in the form of contributions. The operating deficit for 1949 was thus $72,048, compared with $84,658 in 1948. Treasurer Fletcher pointed out that the large increase in publications expense was an investment in better magazines for AIME members, which in future years should bring in more advertising revenue. The preliminary budget for 1950, which Mr. Fletcher also offered, indicates a balance in income and expenses because of a substantial increase in income expected because of the rise in dues and a larger volume of advertising. This can be more accurately established when the final budget is adopted at the April meeting of the Board. The Petroleum Branch, in particular, outlined a plan whereby it might be possible to reduce printing costs and increase advertising income in 1950.

A Controller for the Institute, in the person of George I. Brigden, without salary, as recommended by the Executive and Finance Committees, was appointed to review Institute accounting procedures.

A report of the ECPD Committee on Professional Recognition, recommending that all of the Founder Societies adopt uniform grades of membership, was discussed, but in view of the work involved in reclassifying all present Institute members it was voted not to adopt the proposal.

Considerable discussion arose from a proposal from the Mineral Economics Division, offered by Elmer Pehrson, that the AIME policy on controversial matters, as printed on page 85 of the 1948 Directory, be considerably broadened. Various arguments for and against were offered. The matter was considered so important that action was postponed until the April meeting of the Board to give time for further study.

At a previous Board meeting it had been voted that all Institute, Divisional, and Branch meetings be self-supporting so far as the AIME treasury was concerned, the idea being that no part of a member's dues should be used to finance a meeting which he might not be able to attend. Definite rules for financing and apportioning the cost of such meetings were adopted by the Board at this meeting and copies will be sent to all those concerned.

The plight of young men who had been in World War II and had subsequently completed their education, but who could not become Junior Members because they had passed the 30-year age limit, was discussed by Mr. Head. He advocated granting them permission to be admitted as Junior Members for a period up to three years following granting of their Bachelor's degree, provided only they had not reached the age of 33. Notice was given of an appropriate change in the bylaws to permit this action.

Edward L. Clark, State Geologist for Missouri, presented the essentials of a project now under way by the Engineers Joint Council Committee on National Water Policy, in which AIME participation was desired. The aim of the committee is to give engineering advice to a Presidential Commission that is studying the question. Some difference of opinion was expressed as to the advisability of participating in the work of this committee. Further, it was estimated that the cost to the AIME would be several hundred dollars. It was finally voted to defer action.

A desire to make the Board of Directors more truly representative of all geographical and professional interest groups, and to improve its functional efficiency, led to the appointment of a committee consisting of W. M. Peirce, chairman, E. R. Price and Lloyd E. Elkins, which reported at this meeting. It recommended that for the time being the number of elected Directors continue at 27, with the Divisional Chairmen Directors ex officio. It further recommended that

the Nominating Committee be reduced from 19 to 18. Three of these would be appointed by the President, as in the past. Eleven of the remainder would be chosen by the Council of Section Delegates. Of the other four, two would be chosen by the Mining Branch, and one each by the Metals and Petroleum Branches. Heretofore, the Local Sections have named all members of the Nominating Committee except the three selected by the President.

Of the 27 elected Directors, the Committee felt that the President and past Presidents should not be considered as representing any geographical district, leaving 24. Further, it was believed desirable that the Nominating Committee each year have three Directorships (for a three-year term) at its disposal which are not preempted by fixed apportionment. This leaves fifteen. Of this number, it felt that six should be residents of the New York metropolitan district in order that they can be readily available for the day-to-day work of the Institute, and so that a good representation of Directors may be had at New York Board meetings. Of the remaining nine, it was recommended that one should come from each of nine other geographical districts, which have been delimited to secure an approximate equal number of members in each. The Local Sections included in these districts are as follows:

1. New York.

2. Boston, Connecticut, Pennsylvania Anthracite, Lehigh Valley, Philadelphia.

3. Washington, D. C., Southeast, Florida, St. Louis.

4. Pittsburgh, Cleveland, Ohio Valley, Central Appalachian.

5. Chicago, Detroit, Upper Peninsula, Minnesota.

6. Colorado, Wyoming, Utah, Montana, Black Hills.

7. San Francisco, Nevada, Oregon, Columbia, North Pacific.

8. Southern California.

9. Arizona, Southwestern New Mexico, El Paso Metals, Carlsbad Potash, Permian Basin, Southwest Texas, Gulf Coast.

10. Kansas, Tri-State, Mid-Continent, Oklahoma City, North Texas, East Texas, Delta.

When a vacancy in the directorate exists from any of these districts, a nomination will be made by a committee consisting of the Chairman of the Local Sections. It is suggested that they offer a choice of two men, of different Branch interests.

The Committee also suggested that in 1951 the Nominating Committee name not only a President but also a President-elect, and that thereafter only a President-elect be named each year, who would thus have at least one year of service on the Board before becoming President.

The Committee's plan was adopted and the Secretary was directed to draw up the necessary changes in the bylaws to make it effective.

Distribution was authorized to all new Junior Members of the Petroleum Branch of the Robert E. Hardwicke volume on "Antitrust Laws et al vs. Unit Operation of Oil and Gas Pools," through the courtesy of the Seeley W. Mudd Fund. The translation of "Bergwerk- und Probierbuchlein" was added to the list of volumes from which new Junior Members might choose as an added Mudd Fund gift volume. A second edition of "Basic Open Hearth Steelmaking," to be sponsored by the Fund, was also authorized.

Plans for the 1950 Directory of AIME members were discussed. Instead of printing the Directory from the master file cards at Institute headquarters as formerly, postal cards will be sent to all members about March 1, asking them to supply up-to-date information for printing in the Directory. These cards, when returned, will be sent direct to the printer. If cards are not returned, the titles and addresses on record at the Institute will be used instead. Much of the material in the first 115 pages of the 1948 Directory will be omitted, as will the listing of Student Associates.

The Board referred to the Council of Section Delegates the suggestion of the Gillson Committee on Democratization that a grievance committee be appointed to hear all members' complaints, seek out their cause, and do something about their rectification. Subsequently the Council decided that the hearing of grievances was a matter that it might itself assume, and each delegate was asked to be the clearing house for grievances emanating from members in his own local area.

Council of Section Delegates

On the day preceding the Board meeting the Council of Section Delegates had an all-day meeting, as a result of which certain matters were brought to the attention of the Board: (1) The Council voted to change its bylaws so that an alternate would have the privilege of voting, provided the Local Section from which he came certified to the Secretary of the Council three weeks prior to the meeting the selection of such alternate. The change was approved. (2) The Council asked for information on the reapportionment of Directors to geographical districts. The Board regarded that this had been answered in the report of the Peirce Committee. (3) A reduction in the initiation fee was recommended in the case of Junior Members applying for change in status to Associate Member or Member. It was suggested that one dollar per year of dues-paying membership be credited, up to a limit of $10. This was referred to the Admissions and Finance Committees for study and report to the Board. (4) The Council suggested that a committee be formed from the most influential members of the Institute to approach heads of companies from whom advertising in the Institute's journals might be secured. This aroused a great deal of discussion, which brought out the need for additional funds if all of the technical papers of merit presented to the Institute are to be published. A transcript of the discussion will be prepared for the Finance Committee to study.

The Council commended the Institute staff for the excellence of the three new journals, and President Young for his excellent and indefatigable services to the Institute during his term of office.

Following are the new officers of the Council of Section Delegates: Gerald M. Cover, Cleveland Section, chairman; M. I. Signer, Colorado Sec., vice-chairman; T. B. Counselman, N. Y. Sec.,

secretary; Jack M. Shepherd, East Texas Sec., L. M. Scofield, Minnesota Sec., F. W. Strandberg, Montana Sec., and F. T. Moyer, Washington, D. C. Sec., executive committeemen.

Bylaws of the Minerals Beneficiation Division were amended to provide for a symposium committee, and to require securing acceptance of nominations from the nominees. Bylaws of the new Florida Section were approved, as were changes in the bylaws of several other Sections.

Raul de la Peña informally presented an invitation for the AIME to join with the GSA and the SEG in holding a meeting in Mexico City in 1951, and an invitation was presented by the Boston Section to hold the Annual Meeting of the AIME in Boston in 1953. Action was deferred in each case, the Board expressing its appreciation of the invitations.

A military indoctrination conference will be held in Washington, D. C., in May, and the AIME is invited to send some twenty of its members thereto. Much confidential information as to the military situation, of interest to engineers, will be imparted. Travel expenses to Washington will not be paid. The Secretary's office was asked to receive and forward the names and addresses of all members who would like to participate.

J. Terry Duce and C. Will Wright were approved as AIME representatives on the EJC Committee on International Relations. T. S. Washburn was appointed representative on the Washington Award Commission for a term of two years. Reports on the recent property valuation convention in Lima, Peru, were received by the Board from A. R. Merz and Manuel B. Llosa A., who were AIME delegates.

Officers and Directors Elected

At the executive session of the Board, held at 5 p.m. Feb. 13, C. V. Millikan was elected vice-president to succeed D. H. McLaughlin for a one-year term ending in February 1951; Earle E. Schumacher was elected a Director for a one-year term ending in 1951; and Hjalmar W. Johnson was elected a Director for a two-year term ending in February 1952 to succeed William E. Brewster, deceased. Andrew Fletcher was re-elected Treasurer, and Edward H. Robie Secretary, each for a one-year term. The following were named on the Executive Committee: D. H. McLaughlin, chairman; Earle E. Schumacher, vice-chairman; Philip Kraft, E. C. Meagher, and W. M. Peirce. The following comprise the Finance Committee: Erle V. Daveler, chairman; A. B. Kinzel and Gail F. Moulton.

The following dates for meetings of the Board and the Executive and Finance Committees were approved:

March 15, Executive and Finance Committee, New York
April 19, Board, New York (may be changed to April 21)
May 17, E&F Committee, New York
June 21, Board, New York
July 19, E&F Committee, New York
August 16, E&F Committee, New York
August 29 or 30, Board, Salt Lake City
September 20, E&F Committee, New York
October 18, E&F Committee, New York
November 15, Board, New York
December 20, E&F Committee, New York
January 17, 1951, E&F Committee, New York
February 18, Board, St. Louis.

MBD Sidelights

IN the Annual Meeting photo story, appearing in the March issue, we gave only cursory attention to a dignified ceremony which transpired during the MBD luncheon. Messrs, T. B. Counselman, J. F. Myers, and E. H. Rose were awarded honorary degrees—the first given in AIME history, we believe. The Latin citation is freely translated as follows:

Know you all (Southern Latin) by these presents that we, the undersigned, in the name of the Minerals Beneficiation Division of the American Institute of Mining and Metallurgical Engineers, by this document, confer on

[T. B. Counselman]
[J. F. Myers]
[E. H. Rose]

the first prophet, the voice crying in the wilderness, "Hey, you guys in New York, we want a division," in recognition of extraordinary services, the degree

MILL GENTLEMAN OF DISTINCTION (M.B.D.)

with ten rights and privileges pertaining, which are

(1) To prospect with any and all 'doodle bugs' that guarantee the tonnage and assay in advance.

(2) To teach the Mining Engineer the few simple rudiments involved in delivering a uniform feed to the mill.

(3) To crush all new theories or machines before they spoil your performance record.

(4) To grind to a stop before John Law catches up with you.

(5) To concentrate on redheads at annual meetings, blondes at regional meetings, and the wife at home.

(6) To practice the treating of materials by heat, but restricted to the frying of ham and eggs at Scotch breakfasts, so as not to tread on the toes of other divisions.

(7) To handle all solutions if the value does not exceed a buck ninety-five (imported brands disallowed), including state and federal tax.

(8) To practice operating control herein defined as 'self control'.

(9) Unswerving loyalty to the Division to keep your tail up while keeping mill tails down.

(10) In general, the right to thumb your nose at conventional mill practices while extending your hand to a brother in distress.

For a document to be immortal, it need not be eternal—we affix our hand and seal this 14th day of February, A.D., 1950, in the City of New York.

(Signed) S. J. SWAINSON
Division Chairman

(Signed) DONALD W. SCOTT
Secretary-Treasurer

The *Drift of Things* . . . . . as followed by *Edward H. Robie*

A New Manager of Publications

As all employers know, it is one thing to get a good man and quite another to keep him. When Tom Lippert, was engaged by the Institute as Manager of Publications last July under a two-year agreement it was hopefully believed that editorial, advertising, and circulation problems had been transferred to a head capable of solving them. Changes in the editorial content and appearance of the Journal of Metals and Mining Engineering soon became apparent and by the end of the year advertising was also looking up. In the February issues the first story of Cerro Bolivar—Venezuela's phenomenal new iron mine—made publishing history in the mining field. Then, immediately after the Annual Meeting, Tom was offered a most attractive opportunity in a field completely divorced from publishing—as general manager of the newly organized Titanium Metals Corp. It seemed a chance of a lifetime and the Institute could not well do other than accede to his wish to accept it.

The problem of a successor naturally immediately arose. The possibilities were rapidly surveyed, and the final choice was a man highly recommended by Tom Lippert himself, they having been associated together on the staff of The Iron Age. His name is Tom Lloyd, and he brings to the job much of the same experience as his predecessor, though a few years less of it.

Mr. Lloyd began his duties on March 20, as Manager of Publications and Editor of the Journal of Metals. For the time being at least his sphere will not include the Dallas office.

A new face also appears in the Petroleum Branch office at Dallas where the vacancy caused by the promotion of Joe Alford to Executive Secretary of the Branch has been filled by Jess Adkins, as editor of the Journal of Petroleum Technology.

Too Much Good Stuff

Every magazine publisher endeavors to strike an appropriate balance between editorial and advertising content. This does not mean that he goes to the extreme of turning down advertising, but rather that as advertising increases he publishes more "reading matter". The problem has been somewhat different with the Journal of Metals, which started with only a few pages of advertising and an enormous supply of technical papers, especially in the field of physical metallurgy. For the first year of its publication no brake was exerted on the acceptance of good technical material, but the publishing cost became so large in proportion to the amount received from advertising, subscriptions, and from the dues of members in the Metals Branch that a substantial deficit was incurred. This was especially noticeable because of the new method started a year ago of keeping Branch accounts separate.

For the rest of 1950, a much more rigid editorial selection will be made. Acceptances will be restricted to papers of even higher quality than heretofore. They will be condensed to a greater degree than in the past, thus making them more readable but without omitting essential information. Reviews of the literature in any paper will be more concise, and information of interest only to neophytes will be eliminated. The same data will not be presented in both graphs and tables. Excessively detailed experimental results will be eliminated, the important points being highlighted. Speculation will not be discouraged, but it must be concise. Only really significant micrographs will be used.

We have outlined some of the factors that authors and editors of technical papers in the Journal of Metals have been asked to keep in mind, but the same suggestions apply to all technical papers to be written and published by the Institute. In other words, the meatier they are, and the more readable, the better the chance that they will be published, and the more of such papers that can be published in the course of the year.

Employment for Engineers

A new book entitled "Employment Outlook for Engineers" has just been issued by the Bureau of Labor Statistics of the U. S. Department of Labor (sold by Superintendent of Documents, Washington 25, D. C., for fifty cents). It should be of much interest to college students, educators, young graduates, and to those who employ engineers—in other words, to a substantial proportion of the AIME membership. It is based on data supplied by some 25,000 engineers who participated in the survey. Engineers Joint Council co-operated.

Principal topics discussed are Employment Trends and Outlook, Earnings, and Occupational Mobility.

In the first sentence of the book we learned something: that engineering is the third largest profession, outranked in size only by teaching and nursing. For men, it is the largest profession. It is also one of the most rapidly growing professions and early in 1948 the number of engineers numbered about 350,000, or about three times the combined membership of the AIME, ASCE, ASME, AIEE, and AIChE. This should encourage the membership committees of our societies. By 1960 the number of engineers is expected to be 450,000. Demand for new engineers for the next few years should be 17,000 to 18,000 a year, but in the decade 1940-50 average graduations in engineering will number 19,000 annually. "The number of persons in training is at record levels and the number of graduates expected from these enrollments should more than supply the demands for engineers in the next several years." Last year 44,000 engineers were graduated, so it is evident why some of them had difficulty in finding jobs. Some 47,000 will graduate this year, so job finding may be even more difficult, but the number may drop to 36,000 next year and to 29,000 in 1952. Even these lower figures seem considerably greater than the demand is likely to be, though it is expected that in five years or so demand and supply will be in

balance. Mention is also made that men with special ability or training in research and design are likely to fare better than the average.

What should the schools do? Raise the standards of entry and use more exacting selection methods. Discourage the marginal student.

As to earnings, they average the highest for those with the greatest amount of formal education. "Masters" do slightly better than "Bachelors" and "Doctors" earn considerably more, but this may be because the better men go after the higher degrees. It seems to make but little difference whether one works for private firms or for the Federal Government, but state and local governments seem to pay less. Top-grade men do exceedingly well in engineering, but an engineering training is no guarantee of success for the mediocre. Unfortunately, 50% of the graduates will always be in the lower half of the class, but raising the level of the entire group will eliminate most of the mediocre.

Engineers do a lot of moving around from one job to another. That is well known as a general principle but here are presented data. AIME headquarters can attest to the fact, for our change-of-address desk processes about half of the membership every year. What is perhaps not so well known is the degree to which engineers change from one kind of engineering to another —such as from mechanical to mining. In 1946 more than 20 percent of all employed engineers were working in a field different from the one in which they were trained. Of mining and metallurgical engineers, 27 percent were working in other fields, the largest shift of any engineering group.

Chemical engineering is growing faster than the others. Civil engineering seems to be showing the slowest growth at present. "Reasonable" growth is expected in mining engineering, including petroleum engineering and metallurgy.

Salaries seemed slightly higher for mining and metallurgical engineers than for any other engineering group, but not greatly so.

All Except How to Make the Bomb

Latest of the scientific and engineering developments of the century is, of course, that of atomic energy. Automobiles, airplanes, motion pictures, the radio, and lastly television have in turn attracted the interest of many young men, but now those who are anxious to get in on the ground floor of the newest scientific marvel are attracted to the development of atomic energy. Older men, not trained in modern physics, have but slight knowledge of what this is all about. For both young and old who want to do some serious reading on the subject before the weather gets too hot we commend a new book of 228 pages published by the U. S. Atomic Energy Commission entitled "Atomic Energy and the Physical Sciences". (Superintendent of Documents, Washington 25, D. C., 50¢.) It is the best review of developments to date, current work that is going on, where it is being conducted, and who is doing it that we have seen. Fellowships and educational opportunities are included. If you are in the market for an isotope, or want to know how to recognize one when you meet it on the street, this is just what you need.

April Board Meeting on the 21st

Owing to the annual meeting of the Homestake Mining Co. in San Francisco on April 18, at which President McLaughlin must be present, and the annual meeting of the Canadian Institute of Mining and Metallurgy in Toronto April 17-19, the meeting of the AIME Board of Directors originally scheduled for April 19 has been postponed until the afternoon of Friday, April 21.

Looking for Suckers

One of our members asks if we can't do something about the flood of brokers' literature emanating from Canada, trying to inveigle people on this side of the line to "invest" in Canadian mining and oil prospects. This has been going on for a long time, so presumably the efforts are not without their reward, to the brokers we mean. Our readers are not the kind that rise to the bait, so there is no use in our warning you, and you, and you. There will always be suckers, and no particular harm would seem to result if some of their money went to Canada, where American dollars are at a premium. A little of it might conceivably actually be used for mining development. The technique of some brokers, however, is to pretend not to ask for money. For instance, here is a letter that starts, "Before I tell you a thing about this amazing proposition, let me say that this letter does not ask you to spend a single penny. Not only that but WE WILL NOT ACCEPT ANY MONEY NOW. All I want from you is an expression of opinion. Will you read about it, **and tell me frankly what you think?**

"The first thing I am going to ask you is—Would YOU like to see $5 grow to $500.00, $50 grow to $5,000.00?" (Note the two zeros after the decimal point; it makes the amount look bigger, and perhaps $5,000.00 might be misinterpreted as $500,000.) "It has been done, as you will see if you read further in this letter. And there is a mighty good chance right now that it may be done again. Time and again through the 18th and early 19th centuries the world was electrified by the spectacular mineral discoveries in Ontario and Quebec." (We think he means the 19th and 20th centuries, but let that pass, especially in view of the fact that a lot of Americans seem to be a year off in their reckoning on the present half century, which would seem to us to end on Dec. 31, 1950 instead of 1949.) Then the writer goes on to tell something about the property, in this case a lead-silver mine. It is particularly valuable because lead is "the only known practical defense against ATOMIC RADIATION." (Lead-sheathed houses should be particularly necessary when the atomic war breaks, and their construction will require a lot of lead.) We could go on at length, but we feel sure our readers can guess the rest. No money is asked in this particular letter: merely a promise to invest $50 in 100 shares of stock provided enough additional promises are received to provide "the necessary monies for working capital to commence the mining of its ore reserves," (to say nothing about supplying a bit of capital to the broker). Such letters, too always mention the name of the engineer who assures a bright future for the property; and you needn't look him up in the directory of his professional society for almost certainly his name won't be there.

Arizona Section Holds Annual Meeting

Last November in Tucson 298 people registered for the Arizona Section annual meeting at the Pioneer Hotel. Registration started at 9:30 a.m. and dancing ceased at midnight; it was a long, busy day. Three divisional meetings started off at 10, underground mining, smelting, and ore dressing, and at 2:30 mining geology and open pit mining meetings got under way.

The cocktail hour began at 6, courtesy of Eagle-Picher, G. J. Duff, Arizona manager. It proved to be one of those affairs where everybody had a good time. Dinner was served to 347 people at 7 and was followed by a business meeting, P. D. I. Honeyman, Chairman. Albert Mendelsohn was unanimously elected 1950 chairman.

L. E. Young, president of the AIME, as principal speaker, spoke on a subject that was bound to interest his listeners, preservation of free enterprise in America. Mr. Honeyman then gave a short address and thanked the members and his fellow officers of 1949 for their fine cooperation in making his two years in office pleasant ones. The dance floor was crowded until midnight.

Proposed Amendment to Bylaws

The Board of Directors, at its meeting on February 13, 1950, gave notice of a further amendment to Article I, Section 5, of the Bylaws to be voted on at the April meeting of the Board, as follows:

Substitute for the second and third sentences the following: He shall be not more than thirty years of age at the time of his application, and shall not remain a Junior Member beyond his thirty-third birthday, except that the Board of Directors shall have the power to waive the age limitation in the case of veterans of World War II.

COMING EVENTS

APRIL

3 Boston Section, AIME.
3 Pittsburgh Section, AIME.
4 Washington, D. C., Section, AIME.
4 Society for Applied Spectroscopy, New York City.
4-7 Nat'l. Assn. of Corrosion Engineers, St. Louis.
5-7 Midwest Power Conference, Sherman Hotel, Chicago.
6-7 North Pacific Section, AIME, hosts to the Industrial Minerals Div. of the Pacific Northwest, Mines Bldg., Univ. of Wash., Seattle.
5 Chicago Section, AIME. T. L. Joseph on problems in iron smelting.
7 Columbia Section, AIME.
10-12 Open Hearth Conference, and Blast Furnace, Coke Oven and Raw Materials Conference, Netherland Plaza Hotel, Cincinnati.
10-12 American Crystallographic Assoc., Penn State College, State College, Pa.
11 Delta Section, AIME.
11 East Texas Section, AIME.
12 El Paso Metals Section, AIME.
12 San Francisco Section, AIME.
12 Southwestern New Mexico Section, AIME.
12-14 ASME spring meeting, Hotel Statler, Washington, D. C.
14 Rio de Janeiro Section, AIME.
17 Detroit Section, AIME.
18 Gulf Coast Section, AIME.
19 Southwest Texas Section, AIME.
19-21 ASCE, spring meeting, Los Angeles.
19-22 Electrochemical Society, Cleveland, Ohio.
20 Carlsbad Potash Section, AIME.
20 Utah Section, AIME.
21 Oregon Section, AIME.
21-22 New England Regional Conference, Institute of Metals Division, AIME, Hotel Sheraton-Biltmore, Providence, R. I.
23-26 American Ceramic Society, annual meeting, New York.
24 Alaska Section, AIME.
24-26 AMC Coal Convention, Netherland Plaza Hotel, Cincinnati.
24-27 AAPG, Chicago.
24-27 Soc. of Economic Paleontologists and Mineralogists, Chicago.
24-27 Society of Exploration Geophysicists, Chicago.
25 Montana Section, AIME.
25-26 Annual Metal Powder Show, Book-Cadillac Hotel, Detroit.

MAY

1 Boston Section, AIME.
1 Pittsburgh Section, AIME, Mellon Inst. Auditorium.
1-3 American Geophysical Union, Washington, D. C.
2 Washington, D. C., Section, AIME.
5 Columbia Section, AIME.
9 Delta Section, AIME.
9 East Texas Section, AIME.
10 El Paso Metals Section, AIME.
10 San Francisco Section, AIME.
12 Rio de Janeiro Section, AIME.
15 Detroit Section, AIME.
24-25 American Iron and Steel Institute, Waldorf-Astoria Hotel, New York City.
28-31 AIChE, regional meeting, Swampscott, Mass.

JUNE

6-17 Second Mechanical Handling Exhibition and Convention, Olympia, London.
8-10 National Society of Professional Engineers, Boston, Mass.
16-17 Central Appalachian Section and Coal Division, AIME, Daniel Boone Hotel, Charleston, W. Va.
19-23 American Soc. for Engineering Education, Seattle, Wash.
24-30 First international conference on coal preparation under direction of Charbonnage du France.
26-30 ASTM 53rd annual meeting and 9th exhibit of testing apparatus and equipment, Atlantic City, N. J.

JULY

3-8 Second Conference on Oil Shale and Cannel Coal, Glasgow.
10-15 World Power Conference, London.
30-31 Mining Association of Montana, Hotel Finlen, Butte.

AUGUST

28-31 American Mining Congress, metal mining convention and exposition, Fair Grounds, Salt Lake City.

SEPTEMBER

5-9 6th National Chemical Exposition, Chicago Coliseum, Chicago.
15-16 National Soc. of Professional Engineers, Wheeling, W. Va.
18-23 3rd International Mechanical Engineering Congress, Brussels.
19-23 Conference on Wire Ropes in Mines, Ashorne Hill, Warwickshire, England.
26-29 Assoc. of Iron and Steel Engineers and Iron and Steel Exposition, Cleveland Public Auditorium, Cleveland.

OCTOBER

4-5 Texas Mid-Continent Oil & Gas Assn., annual meeting, Dallas.
4-6 Petroleum Branch, AIME, New Orleans.
9-13 Washington, D. C., Section & Mineral Economics Division, AIME, joint meeting, Shoreham, Washington.
12-13 Petroleum Branch, AIME, Elks Club, Los Angeles.
13 Southwestern Section, Open Hearth Committee, Iron and Steel Division, Houston, Texas.
20 Eastern Section, Open Hearth Committee, Iron and Steel Division, annual all-day fall meeting, Warwick Hotel, Philadelphia.
21-27 National Metal Congress, Chicago.
23-25 Coal Division, AIME, Fuels Division, ASME, Hotel Statler, Cleveland.
23-25 Institute of Metals Division, AIME, fall meeting, Hotel Sheraton, Chicago.
23-27 ASM, Chicago.
23-27 American Welding Society, Chicago.
23-27 Soc. for Nondestructive Testing, Chicago.
27 Southern Ohio Section, Open Hearth Committee, Iron and Steel Division, annual all-day meeting, Deshler-Wallick Hotel, Columbus.

NOVEMBER

3 Pittsburgh Section, Open Hearth Committee, Iron and Steel Division and Pittsburgh Section, AIME, annual all-day meeting, William Penn Hotel, Pittsburgh.
9 AMC, coal division conference, William Penn Hotel, Pittsburgh.
16-18 GSA, annual meeting, Hotel Statler, Washington, D. C.

DECEMBER

7-9 Electric Furnace Steel Conference, Iron and Steel Division, Hotel William Penn, Pittsburgh.

FEBRUARY 1951

19-22 AIME, annual meeting, Jefferson Hotel, St. Louis.

APRIL 1951

2-4 Open Hearth and Blast Furnace, Coke Ovens and Raw Materials Conference, Iron and Steel Division, Statler Hotel, Cleveland.

Among the Student Associates

University of New Hampshire

Students at the University of New Hampshire started off their chapter activities for 1950 on Jan. 11. They saw movies on geophysical methods of prospecting for oil and the development of the wells, once located.

On Feb. 8 they got together again, this time to hear H. Paige, a certified gemologist from Portsmouth, N. H. His illustrated talk covered the various phases of diamond cutting. He concluded his discussion with a brief summary on the criteria that should govern the purchase of a gem. That should come in handy at least once during a lifetime.

University of Nevada

The Crucible Club at the University of Nevada covers its fall meetings in one letter. On Oct. 30, Mrs. H. Foster Bain, whose late husband was Secretary of the AIME, was hostess to the new students at the Mackay School of Mines at her Steamboat Springs home. The turkey dinner and the burro riding will be remembered for a long time.

On Nov. 16, Robert E. Kendall, Mackay School of Mines '48, now a geologist for the Dayton Mine, asked "Is the Comstock Worked Out?"

On Jan. 17, members and guests enjoyed an interesting discussion on the new placer operations of the Consolidated Goldfields and Yuba Dredging Co. at Round Mountain, Nev. L. D. Gordon, chairman of the Nevada Section, AIME, told of the power shovel, the largest operating in the West, and the conveyor belt method of mining the placer ground. Albert Silver, metallurgist, gave the flowsheet of the recovery plant.

Professor W. A. Smythe's announcement that the Woman's Auxiliary of the Nevada Section, AIME, had donated $50 toward the 1950 edition of the "Mackay Miner" was received with enthusiastic applause.

Colorado School of Mines

A really good dinner at the school dining hall got the Colorado School of Mines Student Chapter meeting of Feb. 9 off to a good start. The 46 men present heard V. L. Mattson, director of the Mines experimental plant, report on the activities of the plant and of the newly formed Research and Development Foundation. Speaker for the evening, H. E. Vayo, chief executive of the Personnel Management and Research Corp. of Denver, gave a talk on personnel management which was followed by a vigorous question and answer session on the why's and wherefore's of choosing personnel.

Research Fellowships In Coal and Nonmetallics

The University of Washington offers five fellowships in the School of Mineral Engineering for research in coal and nonmetallics in cooperation with the U. S. Bureau of Mines. The fellowships are open to graduates of universities and technical colleges who are qualified to undertake investigations of research character. Duties start on July 1 or Sept. 1, continue for 12 months with remuneration of $1320 for the year. Subjects for investigation will cover problems in the treatment and utilization of coal and problems in the beneficiation of industrial minerals and ceramic materials.

Applications will be passed on early in May so applicants should submit to Drury A. Pifer, School of Mineral Engineering, University of Washington, Seattle 5, the following information as soon as possible: photograph; copy of his collegiate record from the registrar of his college; statement of his practical and technical experience, if any; statement of his experience or interest in proposed investigations; and letters from three persons covering character, ability, education, and experience.

What Went on at Recent Local Section Meeting

SECTION	DATE	ATTENDANCE	SPEAKER, AFFILIATION, AND SUBJECT
Ajo Subsection	Feb. 9	21	Two movies, Birth of an Oil Field, and Euclids Move the Earth.
Alaska	Jan. 23	35	H. Larsen, Danish National Museum, on archeological investigations in Alaska, and I. Skarland, University of Alaska, on aid of miners and prospectors to the archeologist.
Carlsbad Potash	Feb. 15	60	T. F. Stipp, USGS, on potash deposits in the Salado salt.
Chicago	Mar. 1	73	F. N. Rhines, Carnegie Inst. of Technology, on recent developments in powder metallurgy.
Cleveland	Feb. 23	—	W. T. Rogers, National Tube Co., on statistical methods for quality control in metallurgical work.
Columbia	Feb. 3	24	E. E. Thurlow, USAEC, on pitchblende deposits and other uranium occurrences.
Columbia	Mar. 3	17	H. F. Asmuth, Spokane City Traffic Engineer, on traffic planning; and H. P. Sherman, chairman, Columbia Section, on 1950 annual meeting.
East Texas	Feb. 14	46	J. Payne, Shell Oil Co., on a study group investigation of 20,000 ft drilling.
El Paso Metals	Feb. 8	46	R. E. Fonfer, post engineer, Ft. Bliss, on experiences in Iran.
Kansas	Jan. 31	65	C. Pate, Oil Field Research Labs., on water flooding procedure.
North Pacific	Jan. 19	34	Joint meeting with 23rd annual Mineral Industry Conference. Two movies, and talks by E. T. Hayes, USBM, A. J. O'Donnell, AEC, and A. L. Finlayson, Pacific Car & Foundry.
North Pacific	Feb. 16	43	C. P. Purdy, Wash. Div. of Mines & Geology, on antimony in Washington State.
Pittsburgh	Feb. 8	40	Institute of Metals night. W. M. Ball, Jr., B. Lavin & Sons, on human engineering.
San Francisco	Feb. 24	140	Annual meeting with Woman's Auxiliary at the Mark Hopkins. D. H. McLaughlin, AIME President, Mrs. E. Price, WAAIME Vice-President, and new officers were introduced. Dinner, dancing; entertainment.
S. California, Junior Group	Jan. 12	75	N. Van Wingen, consultant, on well spacing.
S. California, Junior Group	Feb. 9	85	W. T. Cardwell, California Research Corp., on drilling fluid viscosity measurements and their application.
Southwest Texas	Jan. 11	196	L. Elkins, Stanolind Oil & Gas Co., on reservoir studies in a pumper's language.
Southwest Texas	Feb. 8	206	H. Landua, Humble Oil & Refining Co., on developments in wireline type formation testers.
Tri-State	Mar. 1	32	J. Rogers, Univ. of Otago, Dunedin, N. Z., on research at the university, economics in N. Z.
Utah	Feb. 16	87	J. Hall, Chief Cons. Mining Co., on new developments in mine mechanization, and M. E. Volin, USBM, on Afghanistan.

AIME Personals

Charles Asgian is with Party 14 of the United Geophysical Co. at Cortez, Colo.

Robert M. Fleming

Robert M. Fleming became professorial lecturer in mining engineering of the School of Mineral Industries, Pennsylvania State College on Feb. 1. Former general manager of the Ebensburg Coal Co., he has had long experience in mine management, mine safety, labor relations, and other fields concerned with mining engineering.

James S. Baker wrote to us recently saying: "I have just completed an extensive trip through Mexico, Central and South America in connection with our business (James S. Baker Co., manufacturers' representatives) and in the course of the trip I had the pleasure of seeing a number of mining people. **D. C. Deringer, Jr.**, the general manager for one of the Patino enterprises, I saw in Lima. . . . I also saw **George Scholey**. He was a guest of the Japanese at Mukden during the war and is now established with a lead smelter at Medellin, Colombia. The riches of South America are very revealing, and if the political situation ever straightens out, there should be a regular, old time gold rush to some of the countries."

John G. Baragwanath, formerly with the Freeport Sulphur Co., has become vice-president of the Shelter Rock Corp. in charge of its mining and petroleum activities. His headquarters are at 485 Madison Ave., New York City.

G. T. Bator has changed his address from the University of Alabama to the mining department of the Colorado School of Mines at Golden.

Peter A. Bosher completed his course for a B.E. degree in mining engineering at Adelaide University and is now employed as a junior mining engineer with the Western Mining Corp. Ltd. in Kalgoorlie, Western Australia.

George M. Brown, formerly with the Tungsten Mining Co., is now general superintendent of Manganese, Inc., Henderson, Nev.

B. B. Brynelsen is now associated with Noranda Mines Ltd. instead of the Quebec Gold Mining Corp., although his office is still at 1403 Royal Bank Bldg., Vancouver, B. C.

Edward Buchanan has the job of junior engineer with the Calaveras Cement Co. He gets his mail at Box 203, Sutter Creek, Calif.

William E. Bush is with the Berry Holding Co. in Maricopa, Calif. He had been a junior engineer with American Smelting and Refining in Salt Lake City.

William M. Hayden

William M. Hayden resigned on March 1 as manager of industrial relations for the Anaconda Copper Mining Co. to become president of Chemetals Corp., with temporary headquarters at 522 Fifth Ave., New York City. Mr. Hayden has worked directly with **Cornelius F. Kelley**, chairman of the board, on various aspects of world minerals and their impact on our domestic economy through the war and postwar years. He has been with Anaconda and its subsidiaries since 1927.

William S. Cole, Jr., is now reached in care of Reynolds Jamaica Mines, Lydford P.O., St. Ann, Jamaica, B.W.I.

Donald A. Damkroger has graduated from the University of Nevada and is working for the Anaconda Copper Mining Co. His home address is Barrenstein Apt. 5, 510 Montana St., Butte.

Herman J. Daniels, no longer Eastern district manager for the Western Machinery Co., has formed the Daniels Engineering Co. with offices at 519 Markle Bank Bldg., Hazleton, Pa. This firm, comprising a group of outstanding coal preparation engineers, will specialize in heavy media and fine coal work, and will be the official representative for heavy media precision processing in the anthracite field.

Lloyd E. Daume, representative of Southern Minerals and Marketing Corp. Pty., newly appointed consulting engineers for Uruwira Minerals, has his headquarters in Johannesburg, S. Africa.

Edwin H. Davis, president of the New York Coal Co., **C. J. Potter**, president of the Rochester and Pittsburgh Coal Co., and **R. H. Sherwood**, president of the Central Indiana Coal Co., have been appointed Class A directors of Bituminous Coal Research to serve for another two-year term. Class B directors who will be serving the second year of a two-year term include **J. B. Morrow**, first vice-president of the Pittsburgh Consolidation Coal Co., **M. L. Patton**, vice-president of the Truax-Traer Coal Co., and **D. A. Thomas**, president of the Boothton Coal Mining Co.

John B. DeMille, consulting engineer of Montreal, Que., has gone to Rome for six months as a consultant for the ECA, studying Italy's mineral resources.

John B. Dorsh has resigned as editor of the *Mining World* and is working with the Bureau of Mines.

Adrian C. Dorenfeld has resigned from the Copper Queen Branch of the Phelps Dodge Corp., to accept the post of professor of mineral dressing at the University of Alabama's School of Mines. His address is Box 2188, University, Ala.

Eugene A. Eastman, a 1949 graduate of the Michigan College of Mining and Technology, has taken the job of metallurgical engineer with the Tennessee Copper Co. His address is Isabella, Tenn.

Carl Hedman is president of the Hedman Mining Co., Hibbing, Minn., which was incorporated to take over the iron mines operated in

1949 by Hedman & Strand. The Hedman Mining Co. has offices in Hibbing with the Pacific Isle Mining Co. for the operation of the Drew-Syme and Croxton mines on the Mesabi range.

J. C. Elsegood is now engaged in private practice with C. L. Alexander, licensed land and mine surveyor, 22 Pirie St., Adelaide, S. Australia.

M. B. Gentry, vice-president of the Freeport Sulphur Co., has been named vice-president and executive committee member of the American Arbitration Association and assumed his new duties on Jan. 1. He has been with Freeport Sulphur since 1935 and is also a vice-president and director of the Sulphur Export Corp.

Eugene Holman, president of the Standard Oil Co. (N.J.), received the honorary degree of doctor of engineering at the final convocation in commemoration of the 125th anniversary of Rensselaer Polytechnic Institute.

W. G. Hippard can be reached in care of the Liberia Mining Co., Port Monrovia, Liberia, Africa.

D. D. Irwin sailed on Feb. 16 from New York for a visit to the Roan Antelope Copper Mines and Mufulira Copper Mines in Northern Rhodesia. In Cape Town he will join **R. L. Prain,** managing director of these mines, and they will make the trip together, stopping at O'okiep and Tsumeb en route.

Dwight C. Jacoby is a hydraulic engineer with the Bureau of Reclamation of the U. S. Dept. of the Interior. He can be reached at 322 Babb Drive, Oklahoma City 10.

Zay Jeffries retired as vice-president in charge of the chemical department of the General Electric Co. on Dec. 31. Well known in the fields of chemistry and metallurgy, he had been with GE since 1914.

Franc. R. Joubin, formerly in charge of eastern Canadian exploration for the Pioneer Gold Mines of British Columbia, is carrying on a consulting practice from 510-100 Adelaide St. W., Toronto, Ont.

Allen J. Johnson, formerly with the Anthracite Institute, is now available as a fuels and combustion consultant. His office is at 344 Congress Ave., Lansdowne, Pa.

Henry S. Johnson, Jr., is employed as a geologist by the Zonolite Co. of Travelers Rest, S. C.

Fred D. Kay is vice-president of the International Titanium Corp., 120 Broadway, New York City 5.

On Feb. 9 the Moles, the association of leaders in America's heavy construction industry, held its tenth annual award dinner. Pictured above are C. S. Proctor, president of the Moles, Herbert Hoover, who received the nonmember award, Dwight D. Eisenhower, who was guest speaker and was made an honorary member, Richard E. Dougherty, recipient of the member award, and William Durkin, who presented the award to Mr. Dougherty.

George H. Kemmer

George H. Kemmer was appointed mining adviser to the Ethiopian Government last fall and by January was house hunting in Addis Ababa so that his wife and small daughter could join him. Maria Christina, who was born in Milwaukee on Jan. 5, 1949, is a well traveled young lady with two Pacific crossings and her more recent flight across the Atlantic.

Louis Koenig became assistant director of research at the Stanford Research Institute on Feb. 1. He has full administrative charge of Institute activities in chemistry, chemical engineering, metallurgy, applied biology, food technology, and pharmacology. His office was created to provide better and more extensive service in those fields. Dr. Koenig had been chairman of chemistry and chemical engineering at the Armour Research Foundation in Chicago.

W. E. King retired last fall as superintendent of smelting and refining after nearly 34 years of continuous service with the Cerro de Pasco Copper Corp. He started at Casapalca late in 1915 and since 1922 has been at the Oroya smelter organization in various capacities. **J. W. Hanley** became superintendent of smelting and refining upon Mr. King's retirement. **I. L. Barker** is assistant superintendent.

Earl F. Lambert has gone to Baguio, Luzon, P. I., as a junior geologist with the Benguet Consolidated Mining Co.

W. R. Lindsay, who has had a consulting engineering office in Toronto since 1941, has been appointed managing director of Rico Copper Mines. He had been resident superintendent at Anyox and Allenby and general manager for Granby Consolidated.

Iver R. Lerohl has been made assistant chief chemist of the eastern district operations at Virginia, Minn., for the Oliver Iron Mining Co. He had been chief chemist of the Canisteo district and has been with Oliver since 1935. **Esler R. Bechtel, Jr.,** has been promoted to chief chemist of the Canisteo district at Coleraine, succeeding Mr. Lerohl. He also has been with Oliver since 1935. **John M. Martin,** formerly laboratory supervisor at Coleraine, has been promoted to assistant chief chemist of the Canisteo district; he has been at the Canisteo laboratory since 1946.

Federico Luchsinger is general manager of Cia. Venezolana del Diamante, which recently absorbed the Cia. Minera Gran Sabana, at the

same address, Apartado 1862, Caracas, Venezuela.

Wilson T. Lundy, who has played an active role in the American sulphur industry for over a quarter century, has retired as vice-president of the Freeport Sulphur Co. He will continue as a member of the board of directors and in a consulting capacity. He joined the Freeport organization in 1923 and seven years later was made general manager and a vice-president.

J. F. Marley left his post as assistant mine superintendent with Gouverneur Talc at Balmat, N. Y., last October and entered the sales staff of the Eimco Corp., Salt Lake City. He is now located in Eimco's new Duluth branch, making sales-service coverage of the Iron Country.

Stewart M. Marshall, consulting engineer, has moved his office from San Francisco to 748 Bryant St., Palo Alto, Calif.

K. Peters

K. Peters, technical consultant to the Kingdom of Saudi Arabia, has recently returned to his headquarters at Jedda, S. A., from a business trip in the United States and Japan. While in Japan, Mr. Peters made a survey of the industrial and mining centers of that country.

Joseph A. Martino, president of the National Lead Co., has been elected to the board of directors of the Allegheny Ludlum Steel Corp. The two companies have formed the Titanium Metals Corp. of America, jointly owned, for the development, sale, and distribution of titanium metal, its alloys, and allied products.

Robert R. Mates is working for the Combined Metals Reduction Co., Pioche, Nev., engaged in milling operations. The Mates live in Caselton Heights, Pioche, with their son, Daniel Thomas, who arrived on Sept. 2, 1949.

Frank J. McEntee, Jr., is now in Chicago as sales engineer with the Western Precipitation Corp. His address is 4537 N. Greenview Ave., Chicago 40.

Alan Probert

Alan Probert recently resigned from the U.S. Smelting, Refining and Mining Co. after 23 years in the service of this organization and its subsidiaries to return to Mexico for the Bureau of Mines in charge of the cooperative program which the Bureau's foreign minerals region is carrying out with the Mexican Government. This cooperative program involves interchange of methods and ideas between the two countries and is specifically devoted to the development of improved beneficiation practice of complex and refractory ores through the combined efforts of engineers of both governments, using the facilities and laboratories of both Mexico and the United States. Direct administrative control of the Mexican project, as well as all other foreign activities of the Bureau, is under **E. W. Pehrson.**

Donald H. McLaughlin, president of the Homestake Mining Co., and president of the AIME for 1950, has been made chairman of the subcommittee on metal mining matters of the Natural Resources Committee of the U.S. Chamber of Commerce. **Evan Just,** editor of the *Engineering and Mining Journal,* **J. C. Kinnear,** vice-president of the Kennecott Copper Corp., **Lawrence Litchfield, Jr.,** vice-president of the Alcoa Mining Co., **F. S. Mulock,** president of the U.S. Smelting Refining and Mining Co., and **Richard W. Smith,** assistant manager of the Natural Resources Department of the Chamber of Commerce, are also members of the committee.

William Wallace Mein, Jr., vice-president of the Calaveras Cement Co. and the Bishop Oil Co. of San Francisco, has been appointed a member of the National Minerals Advisory Council.

Guinn E. Metzger resigned as metallurgist at the Resurrection Mining Company's mill in Leadville, Colo., last year to go to Alaska. He is an instructor in metallurgy at the University of Alaska, College, Alaska.

Arnold H. Miller, consulting engineer of New York City, addressed the local branch of the Canadian Institute of Mining and Metallurgy on Feb. 10 on avoidance of waste in mining and industry.

H. W. Millett left England for Saudi Arabia in December. He is mill shift boss for the Saudi Arabian Mining Syndicate.

D. D. Moffat is chairman of the western division of the American Mining Congress. He succeeded **Stanley A. Easton.**

L. C. Moore has retired as chief mechanical engineer at Ishpeming headquarters of the Cleveland-Cliffs Iron Co. after 42 years with the company.

George C. Nelms is field superintendent for the Ohio & Pennsylvania Coal Co. Mail reaches him at 333 Spring St., Cadiz, Ohio.

Louis D. Piana, a graduate in mineral dressing from the Montana School of Mines, has been transferred from the Western Machinery Co., San Francisco, to the Ore & Chemical Corp., New York City, as technical adviser for the sale of milling and metallurgical machinery to Europe, North Africa, and South America. He can be reached at Room 3400, 80 Broad St., New York City 4.

Donald D. Phelps plans to work for the Philadelphia and Reading Coal and Iron Co. at Ashland, Pa. for a year before continuing his studies at Penn State.

Good fellowship generated at AIME gatherings hit a new high on the international scale at the Dinner-Smoker at the Annual Meeting, Feb. 13. During the entertainment following dinner, **Enrique J. Piccardo, Jr.,** of Buenos Aires, accepted the challenge of good sportsmanship and warmed the feeling of friendship of the mining fraternity for the people of Argentina as he joined with a group of chivalrous North American caballeros in assisting the Jeannette Hackett Revue in their song and dance act, "Indian Maiden—White Boy." Mr. Piccardo is one of the owners of the only gold mine in the Argentine.

Joseph H. Reid, manager of the titanium division of the National

Lead Co., has been elected a director and member of the executive committee of the company.

John G. Reilly has joined the staff of the American Smelting and Refining Co. as superintendent of the Angangueo unit, Angangueo, Mich., Mexico.

Robert H. Ridgway is in charge of ferrous minerals for the minerals division of the Bureau of Mines at Washington. He had been with the U.S. Vanadium Co.

Paul M. Ripley, of 1944 Grand Ave., Butte, Mont., is an assistant engineer in the engineering department of the Anaconda Copper Mining Co.

O. A. Rockwell has been made assistant general manager of the Calumet division of the Calumet & Hecla Consolidated Copper Co. He has been with Calumet & Hecla since 1943, previous to that having been with Colorado Fuel & Iron and the Montreal Mining Co.

Robert L. Sandvig has completed the Shift Bosses' Training School of the Anaconda Copper Mining Co. and is working in the company's research department. His address is 305½ N. Washington St., Butte, Mont.

Francis M. Smith is working for the Pacific Coast Borax Co., Boron, Calif. He was a shift boss for the Pend Oreille Mines & Metals Co. at Metaline Falls, Wash.

William R. Stern is employed in the research department of the Anaconda Copper Mining Co. His address is 1701 Warren, Butte.

Charles E. Scott, deputy director of the strategic materials division of the ECA, this winter made an extensive trip through Africa, heading a reconnaissance mineral survey in the chief mining districts of Kenya, Uganda, Tanganyika, Northern and Southern Rhodesia, and French Equatorial Africa. Sanctioned by British and French colonial governments, the trip was carried out in cooperation with the respective French governments. **Enzo de Chatelat,** of ECA in Paris, joined the survey in French Equatorial Africa.

"Elements of the Petroleum Industry" Again Available

A third printing of "Elements of the Petroleum Industry", edited by E. DeGolyer, originally published ten years ago, sponsored by the Seeley W. Mudd Memorial Fund, is now available. The price is $2.50 to members of the AIME, including Student Associates, and $5 to others. A new edition has been authorized, but its preparation has not yet begun.

AIME Transactions Volumes

All Transactions volumes authorized for 1948 and 1949 publication are now available except Vol. 182, on Nonferrous Smelting and Refining, which should be published shortly. Copies have been sent to all those on our records as having ordered these volumes; others may place orders at $3 for paid-up members, $6 for nonmembers (domestic) and $6.50 (foreign).

The recent volumes include:

175. Institute of Metals (924 p.)
176. Iron and Steel (552 p.)
177. Coal (594 p.)
178. Mining Geology (556 p.)
179. Petroleum Development and Technology (395 p.)
180. Institute of Metals (713 p.)
181. Metal Mining, Industrial Minerals, Mineral Industry Education, and Geophysics (Rocky Mountain Fund volume). (472 p.)
182. Nonferrous Smelting and Refining (330 p.) (available shortly)
183. Minerals Beneficiation (Gemmell Memorial Fund volume) (372 p.)

Transactions volumes in the new size, 8½ by 11½ in., covering all Transactions material published in the three journals in 1949, are now being issued. Vol. 186, Petroleum Development and Technology (332 p.) is already available. Soon to be issued are: Vol. 184, Mining Branch, and Vol. 185, Metals Branch. The price is $3.50 for a first copy to members, $7 to nonmembers in the United States, and $7.50 to nonmembers in foreign countries.

"Statistics of Oil and Gas Production", with data for 1948 (524 p.) was published last fall. It was supplied free to members who received the Journal of Petroleum Technology. To other members the price is $3, and to nonmembers $6 domestic and $6.50 foreign. Similar data for 1949 will be included in the Journal of Petroleum Technology during 1950, but a bound volume will be available at the end of 1950 or early in 1951, consisting of reprints of the statistical material in various issues of the Journal of Petroleum Technology.

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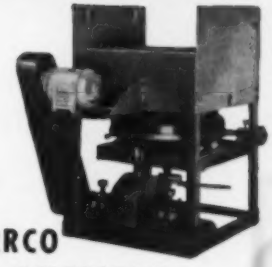

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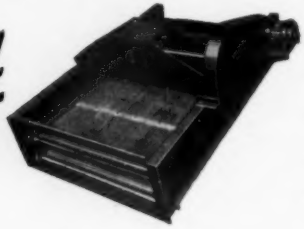

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Obituaries

Frank Drake (Member 1891) died Jan. 14, 1950 in San Francisco; he was 83 years old. Mr. Drake graduated from Michigan College of Mines and also studied at MIT. Before the turn of the century, when such things were possible, he was superintendent of manganese mines in Russia. From 1899 to 1904 he was chief engineer for the Oliver Iron Mining Co. and in 1904 he engaged in investigation of mining and industrial investments.

William C. Hill (Member 1935), engineer with Arctic Circle Exploration Inc., was drowned in Moose Creek near Fairbanks, Alaska, on Sept. 11, 1949. A native of Albion, Iowa, where he was born in 1910, Mr. Hill studied at the New Mexico School of Mines. He worked for the American Metal Co. of New Mexico for a number of years and for about the last ten years was in Alaska with the Cleary Hills Mines Co. and Arctic Circle.

William B. Price (Member 1918), who retired two years ago as director of research of the Scovill Mfg. Co., died at his home in Waterbury, Conn., on Feb. 7. He was 72 years old. One of the country's leading metallurgists, Mr. Price in 1925 developed a new alloy known as admiralty nickel which resists the action of alkalis, hot gases, salt solutions, and inorganic acids. In 1931 he was American delegate to the first International Congress for Testing Materials in Zurich, Switzerland. Following graduation from Yale's Sheffield Scientific School in 1902, he joined the Scovill company as a chemist. A leader in standardizing methods of chemical analyses of copper and zinc, he prepared many technical papers and reports and held patents on a number of alloys in the field.

Selden Scott Rodgers (Member 1913), superintendent of concentrators at the Anaconda reduction department of the Anaconda Copper Mining Co., died on Jan. 8. Born at Toledo, Ohio, in 1886, and educated at Harvard, he went to Great Falls, Mont., in 1910 and worked for Anaconda since then, with the exception of the summer of 1912 which was spent in Alaska. He was president of the Quartz Hill Mining Co. and an active member of the Montana Mining Association.

Robert Ray Van Valkenburgh (Member 1915), eastern states manager for the Globe Union Mfg. Co., Philadelphia, died Oct. 29, 1949. After completing his studies at the Michigan College of Mines, Mr. Van Valkenburgh spent several years in Alaska with the Alaska Gastineau Gold Mines. Back in the States again he worked for the Triangle Mining Co., the Utah Cons. Mines Co., the Wharton Steel Co., and the Empire Steel and Iron Co.

Ernest Waters Wescott (Member 1927), of Comstock & Westcott Inc., Niagara Falls, N. Y., died Jan. 14. Born at West Newton, Mass., in 1888, and educated at Harvard and MIT, he worked as assistant pro-

Necrology

Date Elected	Name	Date of Death
1936	Charles W. Davis	Feb. 21, 1950
1919	Eugene H. Dawson	Mar. 13, 1950
1900	Robert M. Draper	Feb. 18, 1950
1913	Carroll R. Forbes	Unknown
1918	H. W. Gillett	Mar. 3, 1950
1935	William C. Hunt	Sept. 11, 1949
1918	Leonard Larson	Feb. 12, 1950
1902	Tasker L. Oddie	Feb. 10, 1950
1900	Ezequiel Ordonez	Feb. 8, 1950
1927	E. W. Wescott	Jan. 14, 1950
1900	F. W. C. Whyte	Unknown

fessor of chemical engineering at MIT in charge of the Niagara Falls Station and then went on to the Nitrogen Products Corp. He managed Metals Chemical, Ltd., Welland, Ont., for a couple of years and then was put in charge of the Niagara Falls branch of the company which was the predecessor of Comstock & Wescott. Mr. Wescott was also president of Metal Development Corp. and a director of Longyear Process Inc.

James Edwin Ash (Member 1936), who was secretary-treasurer of the Bituminous Producers Board, District 23, Seattle, Wash., died Dec. 17, 1949. Mr. Ash was born at Roslyn, Wash., in 1892, and at seventeen went to work for the Northwestern Improvement Co. In 1912 he became mining engineer for the Roslyn Cascade Coal Co., returning to Northwestern in 1919 as resident mining engineer. From 1920 to 1931 he served as resident and assistant mining engineer with the Pacific Coast Coal Co. in Seattle, becoming superintendent of its New Black Diamond mine in 1931 and of the Carbonado mine in 1934.

Howard Grenville Bohlin (Member 1922), former curator in charge of buildings and grounds at the City College, New York, and more recently professor of drafting there, died on Jan. 25, 1950. Dr. Bohlin was in charge of the underpinning and excavating under Lewisohn Stadium as well as the building of locker rooms, showers, offices, and storerooms in the excavated space. He was born in Jersey City in 1891, received an A.M. degree at Columbia and a Ph.D. at NYU. He began his engineering career with the Southern Minerals Co. and then worked for the Big Four Mining and Leasing Corp., Midvale Steel, El Aquila Oil and Transcontinental Oil companies. He was an engineer on the Holland Tunnel, and chief engineer for AS&R in Mexico. In 1929 he became assistant curator at the City College.

W. M. Claypool (Member 1884), of San Bernardino, Calif., died on Dec. 16, 1949, of a heart attack at the age 86.

Carl Erickson (Member 1933), a mining engineer long active in mining exploration in Canada, died at his home in Toronto on June 28, 1949. Mr. Erickson was 56 years old, a native of Michigan, and a graduate of the Michigan College of Mines at Houghton. He was assayer at Noranda in the early stages of that Company, and afterwards manager of the Amulet Mine before its amalgamation with the Noranda and the Waite. Latterly he was engaged in exploration work

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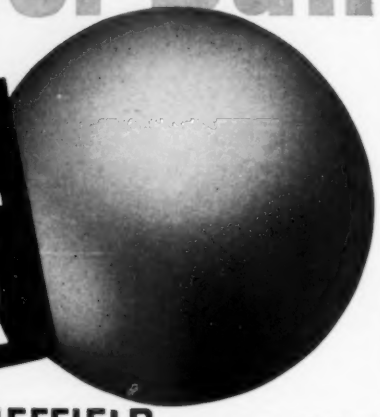

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in the Canadian North West, and Northwestern Quebec. At the time of his death he was head of an exploration firm bearing his own name, the Carl Erickson Associates.

Charles William Faris (Member 1949), partner in Faris & Warren, consulting petroleum engineers, died in August 1949. Mr. Faris was born at Washington, D. C., in 1910 and studied petroleum engineering at Penn State. He worked for the O. R. Seagrams interests for a while and in 1936 joined Shell Oil as a junior exploitation engineer, becoming chief exploitation engineer in 1946. He and his partner opened their consulting office in Midland, Texas, in 1948.

Harvey Lawrence Garrity (Member 1940), recently made general superintendent of operations for the Kennecott Copper Corp. in Utah, died Jan. 11, 1950, in Salt Lake City. Mr. Garrity was born, educated, and worked in Utah; Salt Lake City was his birthplace in 1902; he studied at the University of Utah, and in 1925 went to work for Utah Copper as safety engineer, becoming general track foreman in 1930. After four years with the Bingham & Garfield Railway Co. he returned to Utah Copper as assistant mine superintendent. In 1947 when Utah Copper became the Utah Copper Division of the Kennecott Copper Corp. he continued with the company to become general superintendent of operations.

Walter A. Janssen (Member 1934), consulting engineer of Washington, D. C., died Dec. 17, 1949. Born at Davenport, Iowa, in 1884, he graduated from the University of Wisconsin in 1907 and went to work for the Bittendorf Co. in Davenport, eventually becoming general superintendent. As operating manager of Canadian Steel Foundries Ltd. he was in Montreal for a couple of years. He had served as a director and vice-president of the American Foundrymen's Association. For four years before opening his consulting practice he was vice-president in charge of operations of the American Steel Foundries. He was also with Pittsburgh Malleable Iron & Steel Foundry and the International Vanadium Corp. In 1933 he became deputy administrator of the metals and mineral division of the NRA.

George Martin Krauss (Member 1946), special engineer with Glen Alden Coal Co., died Nov. 12, 1949. He was responsible for the supervision and maintenance of mechanical mining and loading of the company. Mr. Krauss was born at Gelsenkirchen, Germany, in 1900 and graduated from the Clausthal School of Mines. When he came to

the States he worked briefly for a coal company in New York City and then went to Scranton to work for Glen Alden. Mr. Krauss was very active in church affairs, enjoyed the Coin Collectors Club, his stamp collection, and bowled occasionally. He was held in high esteem in his community.

Earl Howard Moffett (Student Associate 1948) was killed in an Army plane. He expected to graduate from Texas College of Mines in 1950.

Thomas James Newton (Member 1944), president and general manager of the Coal Co., Kaska Colliery, Middleport, Pa., died Dec. 19, 1949. Mr. Newton, a native of Pennsylvania, born in Avoca in 1893, worked for many years with the Pennsylvania Coal Co., with time out to serve in the Navy during the first World War. In 1929 he began an association of fourteen years with the Adonizio Coal interests.

Ezequiel Ordoñez

AN APPRECIATION BY CARL FRIES JR.

Ezequiel Ordoñez (Member 1900), internationally known Mexican geologist and mining engineer, died at his home in Mexico City on Feb. 8 at the age of 83. Born in the small town of Lerma in the State of Mexico, Don Ezequiel, as he was known to his friends, attended the National University of Mexico where he specialized in geology, mining engineering, and metallurgy. On finishing his schooling, he joined the staff of the Geological Institute of Mexico, of which he later became Director, and broadened his knowledge of geology by devoting his time to petrology, petroleum geology, and ore deposits. In 1904 he located the first productive oil well in Mexico, at a time when there was general scepticism over the possibility of finding commercial oil deposits in that country. His broad interest in geology was further evinced by his later devotion to mineral exploration, stratigraphy, tectonics, physiography, and volcanology.

In recent years he dedicated a part of his time to teaching and counselling students in geology, besides acting as consultant to numerous private companies, Mexican governmental agencies, and scientific institutions, of several of which he was honorary member or president. His most recent keen interest was in the new volcano Paricutin, which he studied continuously from early in 1943.

Don Ezequiel will long be remembered by geologists of this Hemisphere, not only for his numerous published contributions to geology, but also for his broad vision and for furthering international understanding and cooperation.

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Gordon Arthur Peacock (Member 1937), assistant superintendent of mines at the Ray Mines Division of the Kennecott Copper Corp., died Nov. 30, 1949. He was 43 years old. After graduating from the University of Illinois in 1928 he started his professional career as a mining engineer with the Nevada Consolidated Copper Co. at Ray. In the early 1930's he was with the Illinois State Highway Department and the Museum of Science and Industry, leaving for Rancagua, Chile, in 1934 to be shift boss at the Teniente mine of the Braden Copper Co. He returned to Ray, Ariz., as assistant general mine foreman in 1938 and remained with Kennecott until his death.

Charles Franklin Potter (Associate Member 1906), who retired from professional work in 1935 after 52 years of service, died last December. Mr. Potter was admitted to the New York Bar in 1883 and proceeded to acquire special training in mining law and mineralogy by active practice in Arizona, California, Colorado, Oregon, Wyoming. He was an officer and director of a number of mining companies.

Hiram Smith Rankin (Member 1937), senior mining engineer for the TVA at Knoxville, Tenn., died Oct. 2, 1949. He was born in 1887 at Flint, Mich., and was graduated from the University of Wisconsin in 1909. He went to work for the Oliver Iron Mining Co. and since then had been associated with the Shenango Furnace Co., the Webb Mine, and the Woodward Iron Co. In 1934 he became associate mining engineer in the geology division of the TVA.

James John Rogers, Jr. (Member 1949) was killed in an airplane crash on Dec. 22, 1949. Mr. Rogers was a geologist at the Titanium division, MacIntyre Development of the National Lead Co. at Tahawus, N. Y. He was born at Schenectady, N. Y., in 1925 and received his B.S. degree from the University of South Carolina in 1946.

Franz Nikolaus Rupprecht (Member 1948), special engineer with the Glen Alden Coal Co., died Dec. 29, 1949. Mr. Rupprecht, who was born in Landeck, Silesia, Germany, in 1890, worked for the Otis Elevator Co. for a couple of years and in 1911 joined the Delaware and Hudson Co. He went to Scranton, Pa., in 1918 as surveyor, stripping engineer, and transportation engineer for the Hudson Coal Co. and seven years later began his association with Glen Alden.

A. A. Steel, mining engineer, fruit grower, and flower enthusiast, died Nov. 26, 1949. Mr. Steel earned his E.M. degree after only a year at

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Columbia; he possessed the amazing faculty of accurate judgment of the mineral potentialities of an area. He worked for the Fernando Mining Co., Pacific Consolidated Copper Co., and the Engineering Co. of America. He was professor of mining at the University of Arkansas and worked for the USGS. When his health failed, he went into his father's fruit business, applying scientific methods to that field.

Roy E. Thomas (Member 1921), for many years with the Anaconda Copper Mining Co., died Nov. 11, 1949 in Denver. He worked in various copper mines in the States, South America, and Poland, starting off with Anaconda in 1915. After three years in the army and time out for some studying at Montana School of Mines, he became mechanical engineer for United Verde, returning to Anaconda in 1925 as a designer. That same year he went to Chuquicamata with Chile Exploration. In 1927 he went to Katowice, Poland, to work for a subsidiary of Anaconda. Back in the States again, he worked for the Mine & Smelter Co. as well as Anaconda.

Leland Edward Wemple (Member 1916), former president of the Illinois Zinc Co., died Jan. 3, 1947. He had worked for American Zinc, Lead & Smelting, and American Zinc Oxide, and in 1923 became vice-president and general manager of Illinois Zinc.

Gordon Wilson (Member 1902), mining engineer with the Mexican Gold and Silver Recovery Co. as well as other mining companies in Mexico for more than a quarter of a century, died at his home in San Diego, Calif., on June 3, 1947.

Fred L. Wolf (Member 1918), for many years technical superintendent of the Ohio Brass Co., died at Ann Arbor, Mich., Sept. 27, 1949; he was 64. During the late war, he served as chief of the mica, graphite and rare minerals division of the WPB and after the war became a member of a committee appointed by the Technical Industrial Information Committee to report on nonferrous foundries, ingot producers, and pearlitic malleable producers in Germany. After graduating from the University of Michigan, he worked for the Detroit White Lead Works, then went to Cobalt, Ont., as superintendent of the Silver Lode Mining Co. His association with Ohio Brass started in 1911.

Edward Robert Young (Member 1920), with the Climax Molybdenum Co. in Chicago, died June 9, 1949. Mr. Young went to work as inspector and junior engineer of the water works and filter plant project at Cambridge, Ohio, after graduating from the University of Michigan in 1915. He earned his M.S. degree at Michigan on steel casting research and then went to work for the Detroit Steel Casting Co., becoming research engineer before he joined Climax in 1929.

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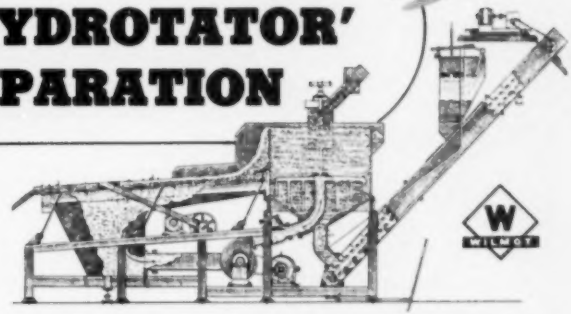

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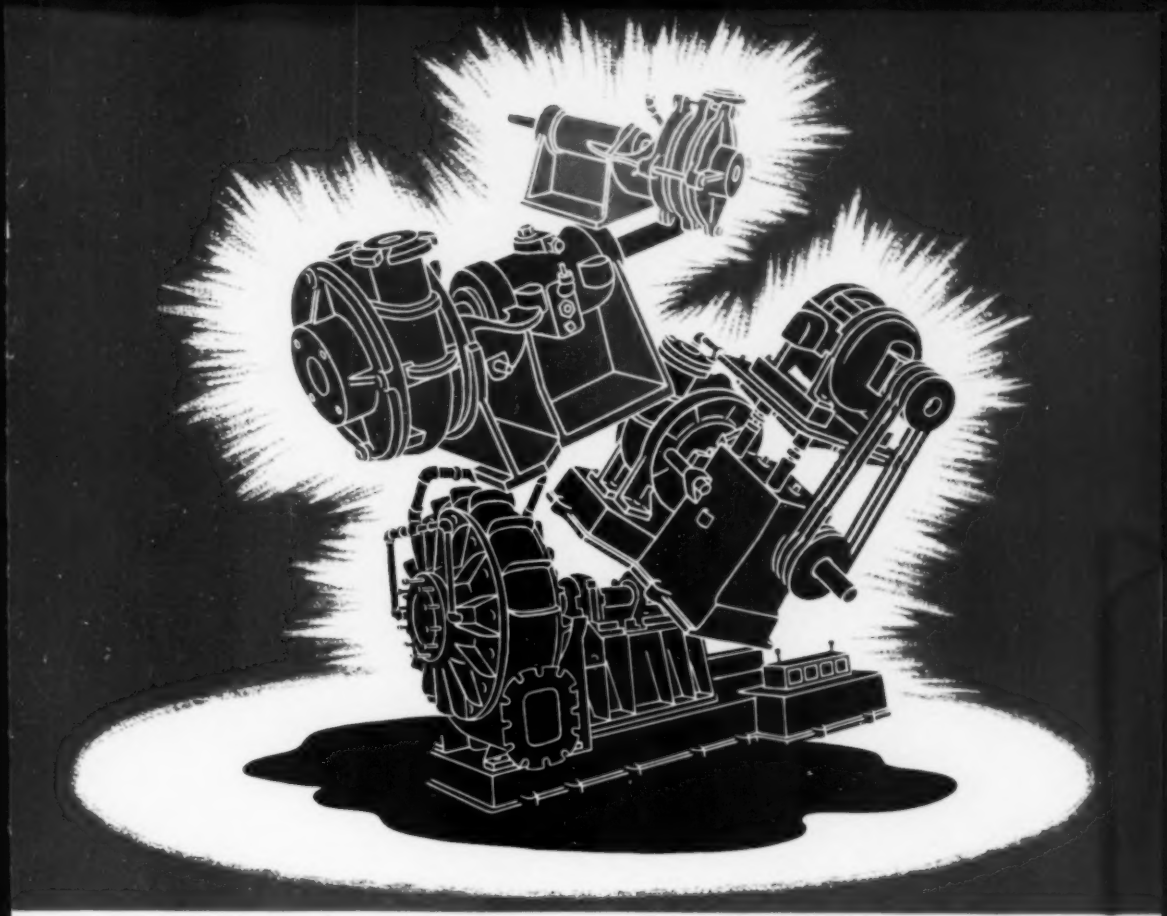

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